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Berthelot

At this place we spoke in our last issue of the loss of Mendeleeff, Moissan and Roozeboom. We have now to record the tragic death of M. Berthelot, a few moments after that of his loved wife. Unlike his compatriot, Moissan, Berthelot was not solely a scientist, but he took a part—and a very active part—in the public political life of France. Yet the grand work of his life related to thermochemistry. The two big volumes in which his experimental determinations of heats of reaction are collected, form a "monumentum aere perennius." They will make the name of Berthelot immortal. It would be useless to dwell any further on the value of his experimental work. But a few words may be said on its relation to thermodynamical theory.

* * *

When we speak of the heat of a chemical reaction, we really mean the energy of the reaction, measured (arbitrarily) in heat units. We can get this energy completely in the form of heat; or we can get it, at least partly, by means of electrochemical processes, in form of electrical energy; or, for instance, by employing expanding gases, we can get it partly in form of mechanical work. In any case the total energy of reaction is the same, whatever may be the way in which we force the reaction to go on. This is the principle of the conservation of energy. Whenever we want to employ this principle—for instance, for giving the heat balance sheet of a chemical or metallurgical process—we need numerical data for the energies of reaction such as Berthelot has determined. This indicates the enormous field of their applicability. But it also indicates the limitations, although Berthelot has maintained a different view on this point for a long time. The principle of the conservation of energy does not state anything whatever concerning the direction in which a reaction will go on. Berthelot, however, believed in the "principle of maximum work"—that if a reaction shall go on at a certain temperature, it is necessary that this reaction shall evolve heat. We know now that pronounced in such generality this rule is wrong. We know now that if we want to find out something on the direction in which a process will go on, we must start from the entropy principle instead of the energy principle; we must know the free energy, not the total energy of reaction. Nevertheless, the strict principle will resolve itself practically into Berthelot's rule for reactions of great intensity and at low temperatures (including our ordinary temperature). It is for this reason that although wrong in its foundation, Berthelot's rule may be usefully employed for many practical purposes.

The Negatives in Storage Batteries

"Many are called but few are chosen." The number of battery inventions and patents is legion. But very few only have marked important industrial progress since the days of Planté. It is, therefore, a pleasure to call attention to one patent of Mr. Joseph Bijur, abstracted in the "Analysis of Current

Electrochemical Patents" in our present issue, since it may be conservatively said that the process described in it looks very promising. It serves the highly useful purpose of preventing the loss of capacity of the negatives (spongy lead plates) in the lead storage battery. For many years the life of a lead cell has been limited by the condition of its negatives. The loss of capacity of the negatives in time during service is a fact. It is a condition, not a theory. As a remedy, negative plates after having lost part of their capacity have been used as positives (peroxide plates) and have thus recovered. But this needs careful and skilled attendance. It is just as if a physician treats his patient at periodical intervals with medicines and thus relieves temporarily the trouble without removing the real underlying cause of the disease. What then is the cause of the trouble with the negatives?

* * *

The chemical reaction which yields the electric current takes place at the contact surfaces of the active-mass particles with the acid in its pores. The active mass with its containing electrolyte is the stomach, heart and soul of the battery. The electrolyte outside of the pores represents only a reservoir of food, from where a fresh supply is dispatched when needed to the place where it is needed, that is, into the pores within the active mass. It is, therefore, an essential necessity that the active mass be thoroughly porous and remain so during service. On the other hand, for simple mechanical reasons, the active mass must be hard, tough and coherent, able to withstand expansion and contraction; it must be firmly attached to the support of the plate so that no particles may drop off or insulating sulphate layers form between support and active mass. These two fundamental conditions—porosity and tough coherency of the active mass—appear to be contradictory, and their simultaneous fulfillment represents an important problem of storage battery manufacture. In the case of the negatives, the loss of capacity may be due to the formation of the insulating sulphate layer mentioned above (especially in pasted plates), or to the dropping off of active material from the grid. But even if care is taken in the construction of the plate to prevent this to occur, there has, nevertheless, always occurred a loss of capacity, due to a contraction of the pores of the mass; the pores contract, the plate shrinks, it loses its porous consistency and solidifies partly into solid lead. This means, of course, an enormous reduction of the active surface, which directly explains the loss of capacity. The problem is, therefore, briefly, to maintain the porous consistency of the plate during service. For plates of the Faure-Brush type experience has shown that it is possible to use a certain composition of the paste, having a tendency rather to expand than to shrink. In an analogous way the problem has been attacked by the European Tudor Co's and the Electric Storage Battery Co. of Philadelphia, which in their "box-plate" employ a very porous active mass of high expanding power and prevent the active mass from dropping off by means of a perforated lead wrapper cast together with the grid. In its results the box-plate meant an important achievement, since with it the negatives have a longer life than the positives.

* * *

Mr. Bijur attains the same end in a peculiarly direct manner. He reinforces the porous structure of the active mass by means of inert matter, just as concrete buildings are reinforced with steel. His plate has a solid metallic support as foundation, and upon it is built a porous structure of particles of active mass, reinforced by carbon particles which are chemically inert, and serve the sole purpose of holding the active-mass particles in their original condition of porous consistency. The idea is beautifully simple. But the means to attain this end would appear at first sight extremely difficult. As a matter of fact, however, the real remarkable feature of Mr. Bijur's process is the extreme simplicity and directness by which he introduces the reinforcing carbon particles afterwards, after the plate has been formed, by introducing sugar and carbonizing it. It is clear that if too much carbon is introduced in this way the carbon will clog the pores and will make matters worse. To be efficient the carbon particles introduced must be just enough to give sufficient strength to the structure, but they should not reduce in any essential degree the active surface of the lead particles. From the fact that the method has been in practical use for a year it appears that Mr. Bijur has found the exact conditions of accomplishing his end. The method certainly deserves most careful attention.

The Bessemer Steel Process

About 12,250,000 gross tons of Bessemer steel ingots were produced in the United States in 1906, with a small tonnage of Bessemer steel castings, made from the various forms of baby converters. The production of Bessemer steel in the United States has been doubling through successively lengthening periods. There is every reason to believe that the 1906 production will never be doubled. The first year, 1867, for which statistics were gathered showed a production of 2,679 gross tons. In 1875 the production was 335,283 gross tons. From that year it has required the following periods for a doubling of production Three years to 1878, three years to 1881, six years to 1887, eight years to 1895, and finally eleven years to 1906. In the early years new plants were built with avidity. Not all of them were entirely successful. Some were poorly located and others were insufficiently financed. There is no necessity to recount the early history of the Bessemer steel process in this connection. The later history, say from 1890 to the present, is of quite a different tenor. There has been relatively very little of building of absolutely new plants, but an enormous advance in the actual production of given plants. Occasionally new plants have been built, but with the completion last summer of the Youngstown Sheet & Tube Co's plant at Youngstown, Ohio, it is quite possible that the building of entirely new plants has absolutely ceased. Not until now has a really successful steel enterprise abandoned a Bessemer plant, but that spectacle is to be presented within a few months, when the Carnegie Steel Co. tears out its Bessemer department at Duquesne, to make way for the eighteen basic open-hearth furnaces it has already started to build, as an addition to the already large open-hearth department at Duquesne.

* * *

The year 1890 is a good selection for comparison with the achievement of last year, since in 1890 the regular Bessemer process had been well worked out, the Clapp-Griffiths and Robert-Bessemer processes were distinctly on the decline, and

direct metal was not being used regularly, although it came into vogue within a couple of years thereafter. We make, therefore, a comparison of Bessemer converters and actual production in 1890 and in 1906. It is impossible to be certain, at this late date, that every one of the converters entering into our 1890 table was actually in operation during the year, but if any were idle the result is not seriously affected. No doubt, on the other hand, enters into the 1906 table; all the converters which it comprises were in operation practically through the year, with the exception of two 10-ton vessels which began operations in August. The 1890 statistics of production excluded the Clapp-Griffiths production, but included the Robert-Bessemer. We have made a suitable deduction for the production by this latter process, arriving at a production, by regular Bessemer converters, of 3,580,000 gross tons of Bessemer steel ingots in 1890. The Clapp-Griffiths and Robert-Bessemer converters are not included in our table for 1890. There were eleven 3-ton and two 2-ton Clapp-Griffiths converters in that year, with one 5-ton, three 2-ton, six 1½-ton and one 1¼-ton Robert-Bessemer converters, a total rated converter capacity by the two processes of 58¼ tons. For 1906 we have made a suitable deduction for the production of Bessemer steel castings, and have correspondingly eliminated all the small converters used for the production of castings. This cuts the standard Bessemer converter capacity off with the two 4½-ton vessels operated at Columbus, the smallest converters now in use for the production of Bessemer steel ingots.

Standard Bessemer Converters Operative in 1890.

Number.	*Rated Capacity.*	*Total Capacity.*
2	11½	23
16	10	160
2	9	18
5	8	40
10	7	70
7	6	42
19	5	95
6	4	24
7	3	21
2	5	10
1	1	1
77		504

Standard Bessemer Converters Operated in 1906.

Number.	*Rated Capacity.*	*Total Capacity.*
3	18	54
9	15	135
4	12½	50
32	10	320
4	8	32
2	7	14
2	6	12
2	5½	11
2	5	10
2	4½	9
62		647

Summary.

	1890.	1906.
Average rated capacity per converter..	6.55	10.44
Total ingot production in gross tons...	3,580,000	12,250,000
Average output in tons per converter..	46,493	197,580
Average output per ton of rated converter capacity.....	7,103	18.934

Three chief causes have contributed to the increase in output, relative to rated converter capacity, an increase which has amounted to 167 per cent in sixteen years. The first has been the introduction, at nearly all important works, of the direct metal process with the Jones mixer. The second has been improvement in the auxiliary equipment, necessitating less waiting on the part of the converter. The third has been a more general realization of the economy in forcing production to the utmost. With direct metal and a mixer hotter and more uniform pig iron is furnished the converter, permitting the use of iron with lower silicon content. There is the curious spectacle of standard Bessemer pig iron being defined in contracts to-day as to contain between 1 and 2 per cent of silicon, while in the large steel works the metal used averages less than 1 per cent in silicon. The iron which is sold runs much nearer to 1 per cent than to 2 per cent, and would not lead to further purchases if it hugged 2 per cent. Naturally, the supply is more dependable with the mixer than when, as formerly, cupolas were used exclusively. The auxiliary apparatus has been improved at many points. The machinery of the converter itself is better, while soaking pit capacity has been greatly increased; interruptions to rolling are less serious, and the increased soaking pit capacity smooths off the irregularities that still come, to a greater extent than formerly.

* * *

The tendency to push all capacity to the utmost is such a feature of American steel works operation that it need not be referred to in particular as to its influence in bringing up the output of converters. There was curious reading for American steel works managers two or three years ago, when a history of a "Bessemer shop" in England was given to the Iron and Steel Institute. The writer described how the output per converter had been steadily increased, so that the management found it "could get along" with a much decreased number of converters, and the useless excess was dismantled. An American manager can hardly sleep well if he finds his whole plant is balanced, except that the shear could do more work than it is called upon to perform. He wants to bring the other departments up. There is no likelihood of any important Bessemer steel works being built in the United States in the future. As already noted, a really successful Bessemer department is scheduled for abandonment, the first case of the sort. Others may follow. The output of the existing plants may be increased somewhat, but the blowing operations are already conducted with almost clockwork regularity, the blast furnace has been made completely the servant of the steel works, to furnish exactly the analysis of pig iron required, and there is apparently but little more that can be done. The pneumatic vessel may not disappear, but may, on the contrary, increase in numbers. It is being put to the, perhaps, base use of constituting an adjunct to the regular basic open-hearth process, by being made to partially desiliconize and decarburize metal preparatory to more complete treatment, and dephosphorization on the open-hearth. The system has been very successful at Ensley, Ala. Probably the principal factor in the extension of this system will be the rate of growth of the demand for steel. With a very rapid growth the scrap outcome will not be sufficient to reduce carbon by mixture, while with a slower growth of demand scrap will come out in steadily increasing quantities, relative to the demand.

Philadelphia Meeting of the American Electrochemical Society.

As previously announced, the sixth annual and eleventh general convention of the American Electrochemical Society will take place in Philadelphia on May 2, 3, 4. The sessions for the reading and discussion of papers will be held at the University of Pennsylvania.

The local Philadelphia committee is sparing no efforts to make this meeting particularly enjoyable. The president of the Society, Mr. Carl Hering, the secretary, Mr. S. S. Sadtler, and the treasurer, Mr. Pedro G. Salom, are all of Philadelphia. This will be the Society's second meeting at Philadelphia. At the first one the Society was founded in the Spring of 1902.

Among the visits and excursions which have so far been arranged are the following: On one afternoon the buildings and laboratories of the University of Pennsylvania will be inspected. A visit will be paid to the plant of the United Water Improvement Company employing the method of sterilization of drinking water by means of ozone. The ozone is produced by silent electric discharges through air by the Vosmaer apparatus. The new United Staes Mint will be visited where the Wohlwill process of electrolytically refining gold, and the Tuttle process of electrolytically refining silver are in use. These two visits, together with an inspection of the Public Buildings (City Hall) will probably be the programme for the second afternoon, while on the same evening a banquet will be held.

For the third afternoon, excursions will be made to the works of Henry Disston & Sons, the well-known tool steel makers of Frankford. The most interesting modern feature of the works is probably the production of special steels in the electric induction furnace of Colby. The Frankford station of the Philadelphia Electric Co. and the Lardner's Point Pumping plant, all at Tacony, will then be visited. For this excursion a boat has been placed at the disposal of the Society. This excursion will be concluded in the evening by a shad dinner at Gloucester, N. J. For those few who may not know it, it may be said that shad dinners on the New Jersey side of the Delaware belong to the most delightful things which old Philadelphia has to offer to visitors in glorious Spring time.

Other excursions are being planned and the above list is merely given to show the probable course of events. Provision is being made for the entertainment of ladies, there being a great many points of interest in and around Philadelphia, such as Fairmount Park, the old State House, with the Liberty Bell, Carpenter's Hall, Old Christ Church, Franklin's grave, the Betsy Ross house, Old Sweeds Church. Excursions for members or ladies for some or all of these points will be arranged.

Among the papers promised for the meeting are the following:

E. Buckingham—The Present Status of the Thermodynamic Scale.

C. F. Burgess—Rapid Determination of Electrolyte Resistance.

G. K. Burgess—Determination of Fixed Points in Pyrometry.

F. A. J. FitzGerald and F. Forsell—On Carbons for Electrochemistry.

Gustave Gin—Electric Reduction of Titaniferous Iron Ores.

W. J. Hammer—Surface Properties of Aluminium and Zinc.

Carl Hering—A Practical Limitation of Resistance Furnaces; the "Pinch" Phenomenon.

W. McA. Johnson—The Electrometallurgy of Zinc, and its Relation to Present Practice.

A. T. Lincoln—Electrolytic Corrosion of Brass (1906 award for research).

Charles E. Lucke—Relative Cost of Power.

John Meyer—Reply to the paper presented by Mr. E. A. Sperry at the Ithaca meeting on Electrochemical Processes as Station Load Equalizers.

H. J. Parker—The Helion Lamp.

Henry Noel Potter—Bomb Calorimeter for Measuring the Heat of Combustion of Substances Giving Solid Oxidation Products.

C. J. Reed—Changes of Concentration and Migration Velocities.

C. J. Reed—Electrolytic Pickling of Steel.

J. W. Richards—The Work Done in Electrolysis.

E. P. Schoch and Alcan Hirsch—The Electrolytic Deposition of Nickel-Zinc Alloys.

E. F. Smith—Recent Developments in Electrolytic Analysis.

R. C. Snowden—Electrodeposition of Zinc.

S. A. Tucker—Boron Carbide.

D. K. Tuttle—Evolution of Modern Methods of Parting and Refining.

W. H. Walker—Influence of Strain on the Corrosion of Iron.

O. P. Watts—The Action of Carbon on Magnesia at High Temperatures.

J. A. Wilkinson and H. A. Gillett—Polarization Voltages of Silver Nitrate Solutions.

Other papers are expected from Professor C. F. Burgess, Mr. E. R. Taylor, Professor S. A. Tucker and others. Of course, the presidential address of Mr. Carl Hering will also be delivered at this meeting, while arrangements are being made for a special experimental lecture at one of the sessions.

The complete programme will be sent out by the secretary to the members early in April, and will be given in full in our next issue.

The Iron and Steel Market.

The little lingering doubt as to the near future of the iron and steel market should be removed by the continuance of heavy specifications against finished steel products and the purchase of some 60,000 tons of Bessemer pig iron for second quarter delivery at substantially the market for prompt delivery.

The developments do not affect the prospects of the market in the two or three closing months of this year and in 1908. That is still an unknown period, with the prospects remaining that the large increase in blast-furnace capacity discussed in detail in our last report will not be balanced by a sufficient increase in consumptive demand over what has existed recently, and still exists. Until about October or November next, the evidence is strong that high pressure will continue, and the latest developments only go to show that the transition from high pressure to insufficient pressure will be more sudden than had been anticipated.

The National Tube Co. has withdrawn all prices on merchant steel pipe, and will accept new business only for delivery after June 1, subject to such prices as may meanwhile be announced. Independent pipe makers have advanced prices one to two points, and the market on steel pipe is firm on the basis of an extreme discount of 75 and 5 per cent off list, an advance of about $2 a net ton in the month.

The American Sheet & Tin Plate Co. has extended through the third quarter, the period for which it will accept tin plate business at the ruling figure. The independent mills had anticipated an advance, and had been taking third-quarter business at 10 cents rise, guaranteed, as usual, against the price of the leading interest. The guarantee will thus cut off the advance in price written in the contracts.

A flood at Pittsburg broke the record of these rivers on March 15, causing a loss of from two to three days' time to most of the furnaces in the district, between 50,000 and 60,000 tons of pig iron, which loss is carried through steel mill and finishing operations, as blast-furnace production had been the

weakest link in the chain. The breaking of the flood record at Pittsburg was not due to altogether unprecedented meteorological conditions, but rather to the progress of forest denudation and the filling in of river banks by industrial operations. The Federal Government took this matter seriously in hand two or three years ago and has put itself in position seriously to resist further encroachment upon harbor lines.

Pig Iron.

The United States Steel Corporation has bought 11,000 tons of Bessemer pig iron from the Bessemer Furnace Association at $22, valley furnace, or $22.85, delivered, Pittsburg or Cleveland, and has taken an option on 14,000 tons for May delivery, which option is virtually a purchase, as it will, undoubtedly, be exercised. The corporation could well use a much larger tonnage, the significant fact being that it is willing to pay so high a price, possibly as much as $10 a ton above its own cost. The transaction cleans up various lots of early delivery iron, which otherwise might not have found ready sale and might have depressed prices. The position of the leading interest is doubtless that while it deprecated the advance in the pig-iron market to so high a figure, it prefers to support the position once it is established. The Cambria Steel Co. bought from W. P. Snyder & Co., 36,000 tons of Bessemer pig iron for equal monthly deliveries over the second quarter, on terms which amount practically to a sale at a flat price of $22, valley. Furnace interests will certainly not sell any Bessemer iron for delivery prior to July 1 at less than $22, and are likely to ask $22.50 for 1,000 to 5,000-ton lots, and $23 for less than 1,000-ton lots. Their position in asking, for months past, $21.50 for second-half iron is much strengthened, as the market is gradually approaching the second half, and remaining on a $22 basis. Foundry iron is stiffer. Small, prompt lots bring as high as $25, valley, for No. 2, while $23.50 is asked for second quarter. Second-half business is practically untouched in the north, with the strategic position of furnace improving week by week. The Southern pig-iron market shows a slightly better tone. The leading interests have been holding to $18.50, Birmingham, for second half, but there have been sellers, possibly on a speculative basis, at $18, and as it is now more difficult to get this $18 iron, the furnaces may possibly be able to establish their $18.50 position for the third quarter, if not for the entire half.

Steel.

The supply of crude steel in the market is reduced and the tone is stronger. The Carnegie Steel Co. has bought 5,000 tons of billets from the New York State Steel Co., a new interest which is just starting a 200-ton Talbot open-hearth furnace at Buffalo, N. Y., and building a blast furnace which should go into commission about Jan. 1 next. A transaction on 10,000 tons of billets was put through a few weeks ago between these interests. These purchases are of interest in the circumstances. The Carnegie Steel Co. will apply the material on its outstanding billet contracts, on which deliveries have been quite behind. The Jones & Laughlin Steel Co. is understood to have withdrawn as a seller of sheet bars, having booked all the tonnage desired at present. The Carnegie Steel Co. has again named $30 on sheet bars on monthly and quarterly adjustment contracts, this price having been put out first for September last on monthly contracts, and for the fourth quarter on quarterly contracts, the previous price having been $29. The company has canceled many of its contracts of this nature, but still has a large tonnage in force. Nominally, Bessemer billets are $29 f. o. b. mill; open-hearth billets are $30 to $32, mill, and sheet bars $30, Pittsburg or Youngstown.

Rails.

While it is well established that on account of greatly decreased borrowing power the railroads will be compelled greatly to curtail their purchases, the influence has not as yet been seriously felt. In but very few instances have postponements been asked, and cars, locomotives, rails, etc., now ordered will carry the steel industry along for six months or more. The Alabama, Colorado and Illinois rail mills are sold up for the entire year and are considering the opening of rail-order books for 1908 deliveries. The Eastern mills are well sold up. To Feb. 15 the Carnegie Steel Co. had sold of standard rails to domestic steam roads, for 1907 delivery, 446,497 tons, while it sold altogether of this class of business, for 1906 delivery, 454,851 tons. Within a few days after Feb. 15 it has sold enough additional tonnage to make the total exceed the entire 1906 business, while there is of necessity considerable business, in small lots, yet to be placed for this year. Its sales of standard rails to traction lines to Feb. 15, for this year's delivery, amounted to only 73,449 tons, against 156,824 tons booked for the entire year 1906, but the placing of this business is always done later, and there is time yet to equal the 1906 tonnage. To March 15 the total rail bookings of the company for 1907 delivery amounted to 745,000 tons, this being made up approximately as follows: Light rails (under 50 pounds), 40,000 tons; standard rails, domestic steam roads, 500,000 tons; traction lines, 75,000 tons; export, 130,000 tons. The price of standard rails remains at $28, mill, except that Colorado charges a higher price, and Alabama charges $29, mill, for its open-hearth rails. It is predicted that the Carnegie Steel Co. will be able to make open-hearth rails at Youngstown by Jan. 1 next, while the new Gary plant will be making rails by Feb. 1 or March 1.

Finished Steel.

There have been no important price changes outside of the advance in merchant steel pipe already mentioned. Galvanized wire products may possibly be advanced on account of the continued high price of spelter, while there may be another advance in galvanized sheets, relative to black. Some Eastern plate mills have reduced the premiums asked above the regular mill price, charged by reason of the earlier deliveries they can make. Premiums are occasionally paid on other products, above the regular mill prices, which are quoted below. Sheets frequently involve a premium of $2 a ton for shipment within a fortnight, while steel bars in the East are regularly at a premium for early delivery.

Structural shapes, $1.70 for beams and channels, 15 inches and under, angles and zees; $1.75 for tees; $1.80 for beams and channels over 15 inches.

Plates, $1.70 for tank quality, ¼ inch and heavier, 100 inches wide and less; flange, $1.80.

Merchant steel bars, $1.60, base.

Common iron bars, $1.80, delivered Pittsburg; for Western delivery, basis $1.65, f. o. b. Pittsburg.

Sheets, 28 gauge, $2.60 for black, $3.75 for galvanized.

Tin plates, $3.90 for 100-pound cokes.

Toronto Meeting of the Canadian Mining Institute.

The ninth annual convention of the Canadian Mining Institute was held at Toronto from March 6 to 8. Mr. George R. Smith, in his address as retiring president, reviewed recent progress in mining and metallurgy in Canada, especially in the province of Ontario, and notably in the cobalt district. The membership of the Institute has now passed the 500 mark.

A special feature of the proceedings was a discussion, extending over two afternoons, on the mining tax law pending in the Ontario Legislature. Two resolutions were adopted, both asking the Government for most careful consideration of the measure before making it a law.

The development of the cobalt district was discussed in several papers by Mr. W. G. Miller, Prof. C. R. Van Hise and Dr. Robert Bell. Most papers dealt with mining and geological subjects. Two interesting metallurgical papers were on the use of graphic formulæ for metallurgical calculations, by David H. Browne, and on magnetic separation by the Gröndal

process, by P. McN. Bennie. The latter paper will be found on another page of this issue.

The main social feature of the convention was a most enjoyable dinner on the evening of March 7. After the official close of the meeting at Toronto, a party of members, numbering more than fifty, made a two-day excursion to the cobalt district.

Mr. Frederick Keffer, of Greenwood, B. C., is the new president.

New York Meeting of the American Institute of Mining Engineers and the Dedication of the Engineering Society Building.

The ninety-second meeting of the American Institute of Mining Engineers will be held in New York City from Tuesday, April 16, to Saturday, April 20, 1907. The date of the meeting has been chosen to enable members to be present at the dedication and formal opening of the new Engineering Society Building, 29 West Thirty-ninth Street.

The full programme of the dedication exercises, on Tuesday and Wednesday, April 16 and 17, will be found below. The provisional programme for the meeting of the Institute of Mining Engineers on the following days, April 18 to 20, is as follows:

Thursday, April 18.—At the session of the American Institute of Mining Engineers in the afternoon, Mr. H. D. Hildage will present a paper on mining engineering operations in New York City and vicinity. This paper will describe the excavation and tunnel work now carried on by the transportation companies. In the evening of Thursday, at the session of the American Society of Mechanical Engineers, Brig.-Gen. William Crozier will present an address on the ordinance department of the engineering organization.

On *Friday*, April 19, a session of the American Institute of Mining Engineers will be held in the afternoon, which will be devoted to the reading and discussion of professional papers. In the evening an informal smoker with vaudeville will take place in the concert hall of Madison Square Garden for members of the three "Founder Societies."

On *Saturday*, April 20, in the afternoon, an opportunity will be given to a limited number of members to visit the tunnel work which will be described in Mr. Hildage's paper.

The programme of professional papers to be presented at the Mining Engineers' meeting is very long. On account of limitations of space we can only give a list of the papers on *metallurgical* subjects.

H. M. Howe—Piping and Segregation in Steel Ingots.

H. M. Howe and B. Stoughton—Wax Ingots Showing the Effects of Casting.

E. G. Banks—Grinding in Tube Mills at the Waihi Gold Mine, Waihi, New Zealand.

Mark R. Lamb—The Butters Slime Filter.

John J. Porter—The Occurrence and Effects of Zinc in the Iron Blast Furnace.

Robert H. Richards—The Velocity of Galena and Quartz Falling in Water.

H. O. Hofman, W. S. Caypless and E. E. Harrington—The Constitution of Ferro-Cuprous Sulphides.

A. L. Sweetser—Laboratory Tests in Connection with the Chlorination Process.

H. O. Hofman, R. P. Reynolds and A. E. Wells—Laboratory Experiments in Lime Roasting and Galena Concentrate with Reference to the Savelsberg Process.

H. M. Howe, William Campbell and C. W. Knight—The Roasting of Argentiferous Cobalt-Nickel Arsenides.

Woolsey McA. Johnson—The Physical Metallurgy of the Reduction of Zinc Oxide.

E. P. Mathewson—Relative Elimination of Iron, Sulphur and Arsenic in Bessemerizing Copper Mattes.

H. O. Hofman, R. Hayden and H. B. Hallowell—A Study in Refining and Overpoling Electrolytic Copper.

Frank Firmstone—An Early Instance of Blowing-in Without Scaffolding-Down.

Charles Catlett—Barium Sulphate Associated with Iron Ore Found in the Pinar del Rio Province of Cuba.

S. S. Wyer—Gas Producer Power Plant.

The programme for the first day dedicatory exercises of the Engineering Societies Building, on *Tuesday*, April 16, is as follows:

The exercises are appointed to commence at 3 P. M. with music. The meeting will then be opened by Mr. Charles Wallace Hunt, presiding officer, who will make the first use as a gavel of the setting maul employed by Mrs. Carnegie in laying the corner-stone of the building. Rev. Edward Everett Hale, chaplain of the United States Senate, will speak a prayer, and communications will be read from the President of the United States, the President of the Republic of Mexico and the Governor-General of Canada. Mr. Charles F. Scott, chairman of the conference and building committees, will then make an historical address, which will be followed by the acceptance of the building by Mr. E. E. Olcott, president of the United Engineering Society, representing the Founder Societies.

Mr. Andrew Carnegie, the donor of the building, will then make an address, which will be followed by music and an oration by President Arthur T. Hadley, of Yale University, on the professional ideals of the twentieth century.

On the evening of Tuesday a general reception will be held beginning at 9 P. M.

The programme for the second day, *Wednesday*, April 17, is as follows: The meeting to begin at 2.30 P. M. After an introduction by Mr. J. W. Lieb, Jr., chairman of the dedication committee, addresses will be presented by the presidents of the Founder Societies: Dr. Samuel Sheldon, president American Institute Electrical Engineers; Dr. F. R. Hutton, president of the American Society Mechanical Engineers, and Dr. John Hays Hammond, president of the American Institute Mining Engineers.

Greetings and felicitations will then be presented from foreign and national scientific societies and institutions of learning, whereupon Dr. James Douglas, past president of the American Institute of Mining Engineers, will make an address. The John Fritz gold medal will then be presented to Dr. Alexander Graham Bell by Mr. Charles F. Scott, chairman of the John Fritz Medal Board of Award.

Dr. A. R. Ledoux, past president of the American Institute of Mining Engineers, will then present medals for distinguished services to Mr. R. W. Pope, secretary of the American Institute Electrical Engineers; Dr. F. R. Hutton, past secretary American Society Mechanical Engineers, and Dr. Rossiter W. Raymond, secretary American Institute Mining Engineers.

CORRESPONDENCE

Tilting Electric Steel Furnace.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—With reference to the paper by Mr. Roechling in your last issue, page 92, permit me to say that while the information on the results obtained in commercial practice with the induction furnace in Germany is highly interesting, yet the design of the tilting furnace is not as novel as the author appears to believe. It is true that Mr. Kjellin in his Norwegian furnaces used two tap holes. But the induction furnaces built in this country according to the designs of Mr. E. A. Colby (who is the original inventor of the induction furnace) have all been constructed as tilting furnaces. This is, for instance, the case for the one furnace in commercial use at the Disston Works in Philadelphia. A. N. H.

Philadelphia, Pa.

The Structure of Wrought Iron.

By Albert Sauveur.

It is proposed in this short article to describe the structure of wrought iron chiefly through the reproduction of the best and most typical drawings and photomicrographs of that metal which have appeared in various books and other publications. By doing this we shall bring together the work of different students of metallography and show in a forcible manner the certainty with which the structure of iron and steel can now be revealed and the close "check" obtained by independent workers. It is also our aim to illustrate and describe the structure of wrought iron in so simple and clear a manner as to leave a distinct impression of its apearance and significance in the mind of even the casual reader.

For the sake of simplicity and clearness we may, at the outset, consider wrought iron as being practically free from carbon, and we shall also ignore for the purpose of this article the presence of such impurities as phosphorus, sulphur, manganese and silicon. In short we shall consider wrought iron as a mixture of two constituents only, namely, (1) carbonless iron and (2) slag, which it essentially is.

Let us now examine the occurrence of these two constituents as revealed by the microscope.

Ferrite.—Carbonless iron in metallography is called ferrite, a name which suggests its nature. An examination of the accompanying illustrations will show that ferrite occurs in the shape of well defined net works in polished and suitably etched sections of wrought iron. The meshes of these net works represent, of course, sections cut through small granules frequently referred to as "grains" or "crystalline grains." It will be noticed that these grains are roughly polygonal and frequently hexagonal.

The dark net work itself indicates the junction lines between adjacent grains and is made apparent by the etching operation, the acid acting around the grains more deeply than over their surfaces and in this way producing a difference in level resulting in the dark appearance of the junctions.

Each ferrite grain is not a true crystal, but is made up of a great many small cubic crystals, which in any one grain are always oriented in the same direction, but whose orientation changes as we pass from one grain to the next and the polyhedric contour of the grains themselves is due to these interfering small cubic crystals. Because of this difference in orientation of the small cubic crystals of various grains, some of these appear bright under the microscope, while others are dark, as plainly shown in Fig. 7. The reason is obvious; those grains which are so oriented that they reflect the incident light into the microscope will appear bright, while if the small crystals have such a direction as to reflect the light outside the instrument, the corresponding grain will appear dark. By slightly inclining the sample in various directions, some of the grains which were bright become dark, and vice versa, because in doing so we change the direction of the light reflected by each individual crystal and this kaleidoscopic effect affords a conclusive evidence of the accuracy of the explanation offered to account for the dissimilar appearance of the grains of ferrite. In order to produce this difference in coloration, however, it is necessary to etch relatively deeply, a slight etching resulting merely in outlining the junc-

FIG. 1.—MICROSTRUCTURE OF CARBONLESS AND SLAGLESS IRON (AFTER J. O. ARNOLD). MAGNIFIED 400 DIAMETERS.

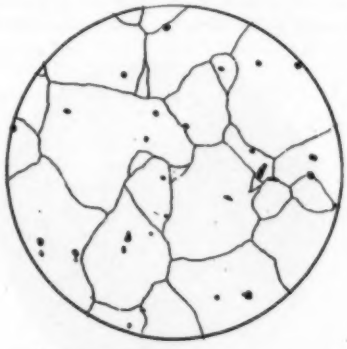


FIG. 2.—TRANSVERSE SECTION OF SWEDISH LANCASHIRE HEARTH WROUGHT IRON (AFTER J. O. ARNOLD). MAGNIFIED 460 DIAMETERS.

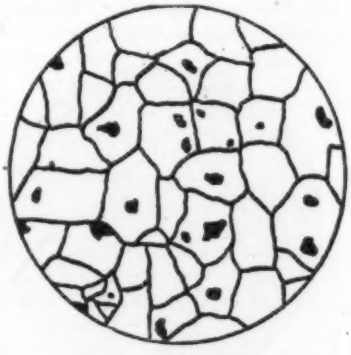


FIG. 3.—TRANSVERSE SECTION OF WROUGHT IRON (AFTER PERCY LONGMUIR). MAGNIFICATION NOT STATED.

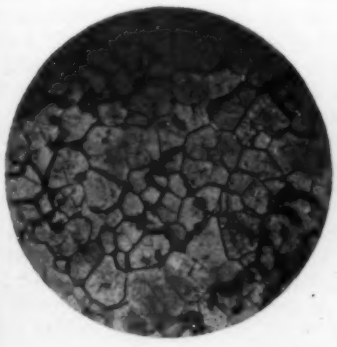


FIG. 4.—TRANSVERSE SECTION OF WROUGHT IRON (AFTER H. C. BOYNTON). MAGNIFIED 80 DIAMETERS.

FIG. 6.—LONGITUDINAL SECTION OF LOW MOOR WROUGHT IRON (AFTER H. C. BOYNTON). MAGNIFIED 80 DIAMETERS.

FIG. 5.—LONGITUDINAL SECTION OF WROUGHT IRON (AFTER PERCY LONGMUIR). MAGNIFICATION NOT STATED.

tions, as it fails to bring out the internal structure of the grains.

Transverse and Longitudinal Sections.—By comparing Figs. 1, 2, 3 and 4 with Figs. 5, 6, and 7, it will be seen that the appearance of ferrite is identical in both transverse and longitudinal sections; there is no indication of the ferrite being drawn in the direction of the rolling or forging. Both sections show equally well the decidedly crystalline nature of ferrite.

Slag.—The illustrations clearly show the way in which the slag occurs in the structure. In longitudinal sections it assumes the form of fibres, varying greatly in width and length, but all running in a direction parallel to the rolling, while in transverse sections they occur as irregular dots, which, of course, correspond to cross-sections through the fibres shown in the longitudinal sections. It will also be noticed, especially in Figs. 8 and 9, that the slag is clearly made up of at least two components, probably two silicates, but whose nature has not yet been ascertained.

FIG. 7.—LONGITUDINAL SECTION OF WROUGHT IRON (AFTER W. J. DEAN). MAGNIFIED 80 DIAMETERS.

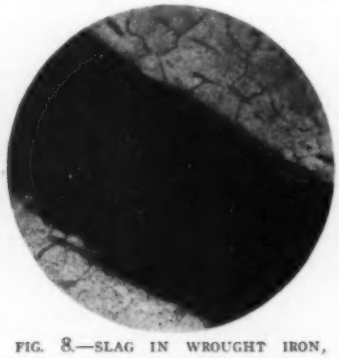


FIG. 8.—SLAG IN WROUGHT IRON, LONGITUDINAL SECTION (AFTER H. C. BOYNTON). MAGNIFIED 80 DIAMETERS.

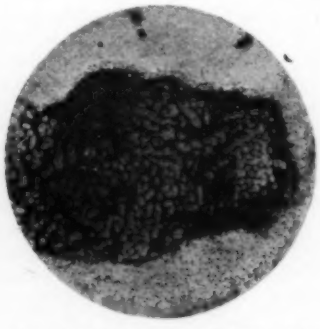


FIG. 9.—SLAG IN WROUGHT IRON (AFTER LEON GUILLET). MAGNIFIED 180 DIAMETERS.

Fibrous Structure of Wrought Iron.—From the above description and accompanying illustrations it will be evident that the ferrite, of which wrought iron is composed, does not assume a fibrous structure, the slag alone being drawn into fibres. Wrought iron, therefore, should not be described as being fibrous, for, leaving aside the presence of slag it is as distinctly crystalline as steel. The general belief in the fibrous character of wrought iron, however, is so deeply rooted that even many modern writers of repute still refer to that metal as being fibrous, and many qualities have been, and still are, claimed for wrought iron because of the presence of these fibres which are denied to steel owing to its crystalline character. It may be that wrought iron resists corrosion better than steel, that it is more weldable and possesses, to a greater degree, other characteristics dear to the blacksmith, but such claims cannot be based upon the fibrous character of the metal and still less, as recently attempted in print, on the presence of iron fibres each one surrounded by a thin coating of slag, a clever, but altogether visionary arrangement.

The structure of wrought iron and therefore its physical properties is affected (1) by the presence of impurities, and (2) by the treatment both mechanical and thermal to which it is subjected. The effect of these factors will be briefly considered in another article.

Blast-Furnace Top Construction.

A most interesting sketch of some modifications introduced into practice in recent years in blast furnace construction is contained in Mr. JULIAN KENNEDY'S presidential address before the Engineers' Society of Western Pennsylvania, which is published in full in the February issue of the *Proceedings* of the Society.

The development of blast furnace construction has been towards much larger sizes of furnaces and towards the use of much wider ranges of iron ore. While the ores formerly used were coarse and lumpy, now large proportions of the Mesaba range ores are also treated which are comparatively fine and dusty, and which have a tendency to cause the furnace to work irregularly and to hang and slip.

When a furnace is working normally the blast pressure drops gradually from the level of the tuyeres up to the zone of fusion, or say 6 feet, above them, where there is a rapid drop for a short distance and then a gradual drop from this point to the top of the furnace. When a furnace is hanging, especially if it stops moving for a considerable time, the stock burns out below, leaving a cavity. In the case of a furnace hanging stubbornly, the interior of the arch formed may look quite smooth, due to fusion going on at this point.

After the cavity reaches a certain size a portion of the stock is liable to drop, breaking the arch and causing the stock above to come down with a rush. At the same time the compressed air in the cavity at the bottom of the furnace is released, and goes up through the falling stock with a tremendous rush. If the top of the furnace is constructed as shown in Fig. 1, having its hopper and cone set on the brick work and held almost entirely by its own weight, this rush of gases is liable to lift the entire top up, or perhaps throw it entirely off the furnace.

It was formerly the practice to set the hopper in a base ring resting on the brick lining alone, but as the lining sometimes disintegrated and allowed the hopper to drop into the furnace, brackets were added to prevent this, as shown in Fig. 1, and later the hopper was bolted down to these brackets or to a frame of beams which took their place. To prevent the upheaval of this bell and hopper apparatus, so-called explosion doors were placed on the down-comers, and in some cases extra openings were carried out through the casings and additional relief doors were fitted to these openings.

In the case of heavy slips, where heavy pressures and large volumes of blast are used, the upward rush of the blast picks up the ore and coke and carries it up with it and hurls it out of the relief doors in great quantities, it being not unusual to see from 30 to 50 tons of ore, coke and limestone thrown out of a furnace in less than as many seconds at the same time that a vast cloud of gases and ore dust envelopes the top of the furnace, and some time these gases light as they come out, making a tremendous flame.

Mr. Julian Kennedy took up, several years ago, the problem of preventing these so-called "explosions" with resulting loss of life and other damage. The fundamental point of his solution is that we have not to do here really with an "explosion." What really happens is that it is simply the compressed air from the blast in the bottom of the furnace rushes upwards and does the damage.

By examining the construction of blast furnaces as used at that time in the United States, Mr. Kennedy was unable to find a single furnace the top of which could not be lifted by a pressure of 2½ pounds per square inch. The throwing off of the top is therefore fully accounted for by the pressure of the compressed air accumulated in the cavity below a handing charge.

Now, it is quite clear that the ultimate pressure which can thereby be produced in the furnace as result of a slip cannot be more than the pressure in the blast main, or, in other words, the pressure which produces the damage comes entirely from the blast engine. Now, a stream cannot raise higher than its source, and if a furnace casing is made otherwise strong enough to resist this pressure it will be entirely safe to reduce the size of the outlets to such an extent as to offer a material resistance to the flow of the gases from the furnace in the event of a sudden and heavy rush of blast through the furnace.

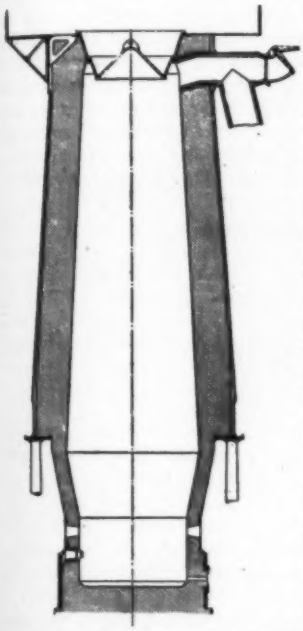


FIG. 1.—OLD CONSTRUCTION OF BLAST FURNACE TOP.

Such a checking of the gases at these outlets will greatly reduce the rapidity of flow for the moment, and will also diminish the tendency to carry solid materials out of the furnace. (In the discussion which followed the paper the following suggestive analogy was pointed out: The throttling of the gases at the outflow will reduce the amount of dust carried by them for the same reason that the waiter who pulls the cork entirely and suddenly out of the bottle will lose at least half the champagne, while the fellow who pulls it gradually will not lose a particle.)

The first furnace built on these principles of Mr. Kennedy was at the Iroquois Iron Co., in Chicago, but there are now in operation twenty-four furnaces of this general construction, and these furnaces have demonstrated, beyond a shadow of a doubt, that it is absolutely safe to altogether close the top of the blast furnace, and that the letting loose of great volumes of dust and lumps of stock at this point is entirely unnecessary.

It has been said that even if the dust is all brought down the down-comers, it will pass through the stoves and boilers, and thus be scattered over the surrounding country. Some of it, of course, will, if no arrangements are made to stop it, but a large proportion of it can be caught in very simple dust catchers, and since gas engines have been installed at furnaces, there have also been started gas washers so perfect that they will take gas from a furnace running on a mixture consisting largely of Mesaba ore and cleanse it till it has less dust in it than the ordinary atmosphere. So that given a device that brings all the gases down to the cleaning apparatus, it is entirely feasible to do away with the dust absolutely.

Mr. Kennedy concluded with the remark that there surely can be no excuse in the future for burning men on top of furnaces, or burning or otherwise injuring them with materials thrown out of the tops of furnaces because of slips, and the amount of dust sent out from a furnace will depend entirely on how perfectly the management choose to cleanse the gas by means of dust catchers and washers, which have at the present time been developed to a degree of perfection which will enable the top of a furnace draft stack to be "cleaner, as far as dust particles are concerned, than the cleanest spot in Pittsburg is at present."

Of course, all the above considerations refer only to the bad results of slips, which are falsely called explosions, and not to real explosions. They are known to have occurred in furnaces when blown out, and when nearly empty a lot of air possibly gets through the thin incandescent bed of fuel without being burned and mixed with the gases above, and then the whole business exploded. These are real explosions.

In the discussion the use of the blast furnace gases was referred to by various speakers. Mr. Kennedy referred to Sir

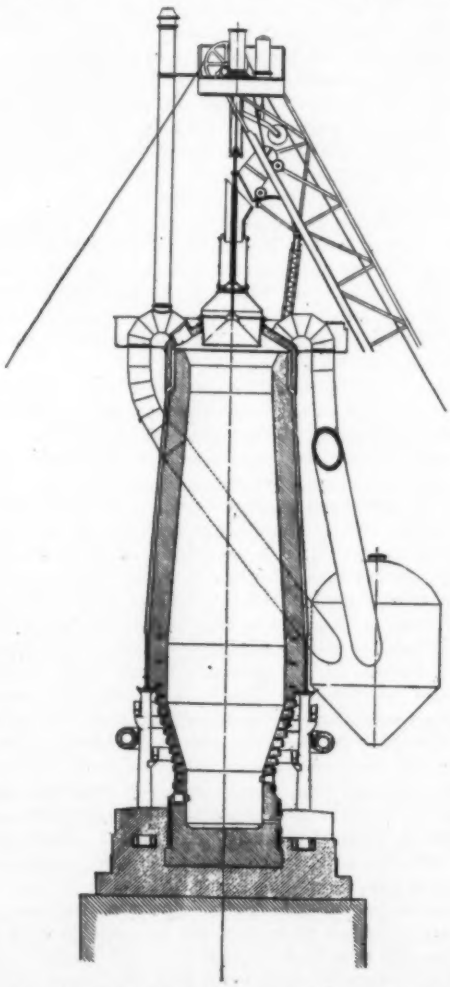


FIG. 2.—MODIFIED CONSTRUCTION OF BLAST FURNACE TOP.

William Ramsay's statement that in the future the blast furnace would be run as a gas producer and pig iron would be a by-product, and thought that this statement is perhaps not so wild as it would at first seem. He expressed his opinion that the most suitable way of utilizing the blast furnace gas is in gas engines.

For other commercial purposes the gas is probably too weak. It has not much illuminating power, and has not heating power enough to use, for example, in open-hearth furnaces.

But the gas is an ideal gas for gas engines, simply for the reason that it is not so strong as some of the other gases.

Compound Steel—A Modern Steel Plant Proposition.

By Peter Eyermann.

Open-hearth steel versus Bessemer steel—this is the great struggle between two rivals in modern steel manufacturing plants. The average user of open-hearth steel claims that this quality of steel is better than the Bessemer. The manufacturer, however, has much less trouble and costs with the latter. Therefore, it is his intention to keep the market up with Bessemer quality.

The complaint has been frequently heard that for many applications this material is less suited than that produced in the open-hearth. From a scientific standpoint it is difficult to see why the latter should be superior, since the chemical end result in both can be made entirely alike. But there is a physical explanation for this practical inequality. The blowing operation is probably too rapid to permit keeping pace with the reactions within the modern large sizes of Bessemer converters all uniformly through the entire bath of molten metal; there is also no time for thorough continuous chemical analyses. And even when the spectroscope is used for observation of the flame, one merely depends on the physical tests and senses of the assayer or men in charge.

Of course, since the general adoption of the mixer, the blowing process, for both acid and basic converters, was so far simplified that any question of a difference between Bessemer and Martin quality could have been entirely disregarded, because the composition of the charged liquid pig iron which is drawn off from the mixer is definitely determined beforehand. But the old prejudice exists yet.

In the open-hearth process, however, the mixer never played such an important part, because there was always ample time for both chemical and physical tests and observations, and the uniform quality was thus assured without resorting to the use of the expensive mixer. But also in this case it is much easier to produce the desired quality of steel by using the mixer, as the pig iron contains always an approximately uniform and even percentage of silicon, carbon, manganese and phosphorus.

At the present time, therefore, in the various methods of steel production the governing factor has come to be not so much a question of quality as one of costs and quantity. The Bessemer process is still unrivaled as regards its cheapness, and the Siemens-Martin process still unrivaled concerning quality and cheapness, the latter only in comparison with the expensive crucible steel. The average production of an open-hearth furnace is yet about 100 tons in 24 hours and 800 tons daily for the Bessemer furnace. The output of Thomas converters is generally less. It may, therefore, be safely assumed that one open-hearth is equivalent to only one-seventh of a converter in point of production, which fact affords an excellent indication of the greater capacity of the converter steel works of to-day.

Now, since human knowledge of iron ore started, since those gray historical times up to date, the idea was never dropped to make steel straight from iron ore. The readers of this journal are, of course, acquainted with this history, and it is not the purpose of these lines to repeat known things. I will mention here only the recent developments of various electrical methods—in my mind all ingenious theoretical experiments, but impractical yet.

Before going into the real subject I may call attention to the recent development of the modern gas engines driven by blast furnace gas. For many years this gas was escaping unseen in daytime, and the bright light from the furnace tops was throwing its shine into the night down on the human buildings, like light fires are guiding the ships on the ocean now. In this way uncounted millions have been and are still being lost. It is only a few years ago that even up-to-date engineers asked the question, "What has a gas engine to do with a blast furnace?" But now so much has already been written on this subject that I will note only the main points to be considered in this question.

The blast furnace of the future will be considered as a big gas producer. The gas might be used in gas engines to generate electricity for general power purposes, or it may be used in gas furnaces for the manufacture of steel or other products. There is already also some talk of the entire abandonment of the coke ovens. But as this field is not yet so far developed, I have to start with those fuel furnaces for the blast furnaces.

The following calculation is of minor influence on my compound-steel furnace, but it is a part of it, as it proves that this coke oven gas is available in sufficient quantities in modern by-product plants so as to allow its further use in steel furnaces or in gas engines. Eleven thousand tons of soft coal yield in average about 10,000 tons of dry coal, and are converted to about 7,850 tons of coke. The balance left consists generally of

110	tons	of sulphate of ammonia.
100	"	tar.
30	"	pitch.
60	"	benzol.
5	"	naphtha.
2	"	sal ammoniac.
120	"	ashes, etc.
1,600	"	coke oven gas.

Practice has proved that 1 ton of coal is equivalent to about 9,000 cubic feet or even more of such gas. The heating value is somewhat lower than that of city gas, averaging, say, 500 B. T. U. per cubic foot. A good gas engine will yield 1 hp. per hour for 10,000 B. T. U. But considering less favorable conditions, 12,000 B .T. U. may be assumed in this calculation; 24 cubic feet per hour of this gas are therefore required per horse-power.

Fifty per cent of all gas has to be taken for heating the coke ovens, and if ten further per cents are deducted for uncontrollable leakage, only 3,600 cubic feet of gas are available for power per ton of coal, or $3,600 \div 24 = 150$-hp. hours for each ton of coal charged in the coke oven.

This result can be checked in the following way: One pound of coke oven gas is equivalent to about 33 cubic feet at normal temperature; 2,000 pounds, or 1 ton of coke gas measures 66,000 cubic feet, hence the above noted 1,600 tons of coke oven gas measures 105,600,000 cubic feet of gas. Sixty per cent off for heating and loss leaves for power 42,240,000 cubic feet, yielding 1,760,000-hp. hours. The quantity of coal charged was 11,000 tons, and $1,760,000 \div 11,000$ gives 160-hp. hours per ton of coal charged.

Practical experience shows that in a blast furnace plant there are required about 8 to 10 hp. per ton of pig iron produced to run the plant. A 500-ton furnace would require, therefore, about 5,000-hp. machinery. It has already been claimed that a coke plant near the blast furnace plant is sufficient to run the latter. Is it true?

Five hundred tons of pig will require, roughly, 500 tons of coke; 1 ton coke equals 1.3 to 1.4 tons of coal. If we consider only 1.3 tons, we have $500 \times 1.3 = 650$ tons of coal charged in the coke ovens in 24 hours. Hence, $650 \times 150 = 97,500$-hp. hours will be available from the coke plant during 24 hours. This means that there are continually available $97,500 \div 24$, or approximately, 4,000 hp. If we assume a power requirement of only 8 hp. for the ton of pig iron, these 4,000 hp. cover exactly the requirements of the 500-ton furnace plant.

If I can cover, therefore, the wants of my blast furnace plant by my coke ovens, why not try to cover the requirements of my steel plant by the blast furnace plant? Is it possible?

Much has been published in recent years on the proposition to use the available blast furnace gas in gas engines, and many practical results have been obtained in Europe, and more recently also in America. I will, therefore, deal with this subject

only in so far as is necessary to find the amount of gas from the blast furnace which is used in the steel plant and will discuss how it is used in a plant.

A considerable number of furnaces are operating now with an average output of 24 tons of pig iron per hour, or 576 tons per day of 24 hours. For convenience in calculation we will use this figure.

With the object of obtaining as much gas as possible from the blast furnace, using it as a large gas producer, the pig iron can be obtained, on paper, at least, as a by-product in modern steel manufacture. Here is the proof:

With the object of obtaining somewhat richer blast furnace gas than may be obtained by using coke alone, some soft coal may be charged into the furnace with the coke. It may be preferable under certain conditions to mix coke oven gas with the blast furnace gas in order to secure a more uniform gas or to install a "mixer" also for all gases of one complete plant.

For each ton of pig iron we get 150,000 cubic feet of gas, having an average heating value of 100 B. T. U. per cubic foot. If, therefore, all gas would be transformed into power, then 150,000 × 100 = 15,000,000 B. T. U. would be at disposition for each ton of iron. The gas engine requires 12,000 B. T. U. per hour to produce 1 hp., so that 1 ton of iron per hour will yield 1,250 hp., or in a 576-ton daily plant about 30,000 hp. will be available. In the near future a good deal of the heat which is required to heat the blast may be obtained from the liquid cinder, thus doing away with the high and costly hot-blast stoves, but at present we must still consider them as follows:

Twenty-five per cent of the gas of the blast furnaces is to be deducted for heating the stoves. The plant producing 576 tons of pig iron daily, yields 3,600,000 cubic feet of gas per hour; 25 per cent of this amount, or 900,000 cubic feet, are to be set aside for heating the blast.

To supply the required amount of air in the blast furnace the blowing engines are built on an average of, say, 80 to 100 cubic feet of air per pound of coke charged in furnace. For 24 tons per hour are therefore required 2,000 × 24 × 100 = 4,800,000 cubic feet of air per hour, or 80,000 cubic feet per minute. With a pressure of up to 25 pounds per square inch, each 100 cubic feet per minute require about 4 to 5 hp. per hour; hence a furnace of 24 tons of iron per hour requires for its blowing engines from 3,200 to 4,000 hp. The modern gas blower requires 12,000 B. T. U. per horse-power-hour, or if we assume again 100 B. T. U. per cubic foot of gas, about 120 cubic feet of gas. This would result in a further deduction of 480,000 cubic fet of gas for blowing engines, but as these engines are not always running under full power, a figure of 400,000 cubic feet per hour may safely be assumed.

The uncontrollable leakage in an up-to-date plant should be not more than 5 per cent, or 3,600,000 × 0.05 = 180,000 cubic feet per hour.

Mechanically-operated gas-cleaning plants have given better results than plain washing towers; for this purpose are required from 0.4 hp. to 0.7 hp. per 1,000 cubic feet gas. If all gas is cleaned (a proposition which has to be considered in a modern plant) we get 3,600 × 0.7 = 2,520 hp., or 302,400 cubic feet of gas. Since this plant is not always running under full load, 252,000 cubic feet per hour will be a fair figure for this purpose.

For auxiliary machinery, such as ore, lime and coke-handling apparatus, electric light plant, power transmission, compressed air, oil and water pumps, elevators and hoists, pig crushers, casting machine, etc., there are required per ton of pig iron per day not more than 2 hp. per hour, or for our 576-ton daily plant a 1,152-hp. engine. Making again some allowance for times of light load, 115,200 cubic feet of gas will be required per hour for this purpose. The balance sheet for gas is therefore:

	Cubic Feet.
Obtainable per hour	3,600,000
Losses, blast heating	900,000
Blowing engine	400,000
Leakage	180,000
Cleaning	252,000
Auxiliary power	115,200
Total loss	1,847,250

Hence there remain for utilization 1,752,800 cubic feet per hour.

Considering 100-B. T. U. quality, this means 175,280,000 B. T. U. per hour for further use in gas engines, which for 12,000 B. T. U. per horse-power-hour means 14,600 hp., or, in accordance with standard gas engine experience, about a 10,000-kw. plant.

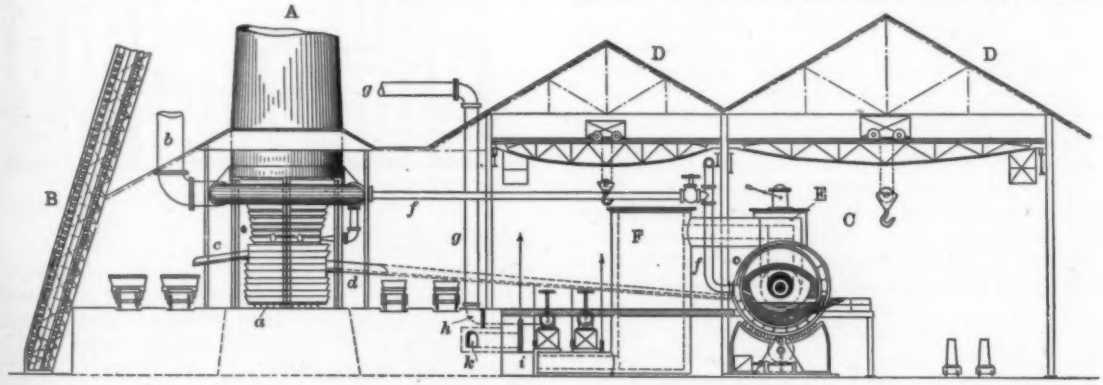


FIG. I.—DIAGRAM OF ARRANGEMENT OF BLAST FURNACE AND STEEL PLANT.

From this we can deduce that the blast furnace as a gas producer covers the costs of manufacturing pig iron. For if we assume an exceptionally high cost for manufacturing pigs at $12.00 per ton, we have in our 576-ton daily plant an expense of $288 per hour. The 10,000 kw., however, we have in surplus, and can sell them, say, for 2½ cents or 3 cents per kw-hour to the public in form of electrical energy. This means an income of $250 to $300 per hour, thus equalizing the entire manufacturing costs.

* * * * * *

From this departure I now return to the steel subject. The main object of my proposition is a new method of manufacturing steel, and this method has the following advantages: (1) The cost of production is reduced; (2) at the same time the tonnage of the output is increased; (3) a less costly plant is required; (4) the plant is independent of the market prices of scrap; (5) less coal is used for the open-hearth quality than hitherto. The writer was very much pleased to find some time ago a similar furnace proposition from another source, working in a large Southern steel plant simultaneously on the same

object. An article on this matter was published in *Stahl und Eisen*, in Germany, by Mr. Goldstein, of Monterey, Mex. I mention this here only to prove that my own standpoint is also based on good practical experience of others.

Fig. 1 shows the general arrangement of my improved steel plant directly attached and operating simultaneously with the blast furnace plant. The blast furnace A may be of any suitable type and construction. My improved type with the all-over-bosh-and-bottom-water-cooling system is shown, however. Like the skyscrapers of to-day, the entire structure rests on a grate of heavy beams *a*. The charging hoist is shown and indicated by B. The hot blast is supplied by line *b*, and the blast nozzles shown, too. Slag notch is at *c* and metal tap at *d*.

There is no mixer required, as in my mind this expensive vessel is in fact only an unwieldy adjunct, the use of which was rendered compulsory solely on account of the inefficiency of old-fashioned appliances and methods adopted to meet modern requirements. The working of my new plant will be effected without the aid of the expensive Bessemer blowing engines, such as required for the converters; a rough fining of the liquid pig iron, tapped directly into the compound steel furnace C by means of cold or hot blast from the *blast-furnace* blowing engines, may be obtained from a pipe connection *f*. Further, the expensive buildings necessary for the accommodation of the long rows of open-hearth furnaces of to-day can likewise all be dispensed with, together with the entire gas producer plants. The building D, as shown, would not take more space than that of the attached cast houses of a modern blast furnace plant.

The furnace C will cost about twice as much as a modern large stationary open-hearth and the same amount as an up-to-date "Talbot" or other big tilting furnace. E represents the apparatus for increasing the quality of the blast furnace gas to be used for heating purposes in the open-hearth. A pipe connection *g* for this purpose may branch off somewhere conveniently from the furnace gas main, and is attached at *h* to the reversing gas valve *i* of the furnace; *k* indicates the flue to any of the furnace smokestacks.

The steel furnace, called "compound furnace" hereafter, on account of the combined open-hearth process and converter method, is shown in Fig. 2 and Fig. 3. Fig. 2 is a plan view; Fig. 3 a longitudinal section, and C in Fig. 1 is a cross-section.

The inclined air—blast nozzles, acting on or in the bath of the metal, are shown in C and in Figs. 2 and 3 as well. The

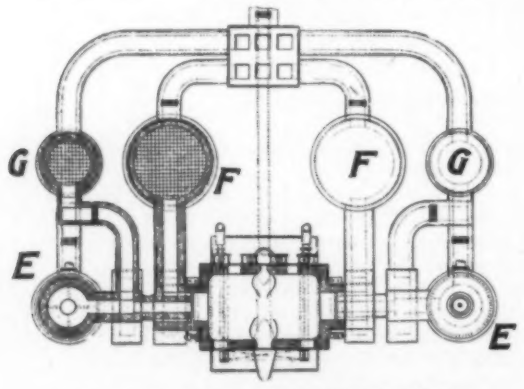


FIG. 2.—PLAN VIEW OF COMPOUND FURNACE.

"improving towers" for gas are E throughout; the gas regenerators, in this case better called "reheaters," are shown at G, and the reheaters for air at F.

As the usual furnaces are too well known to the readers of this journal I will not waste space and time in further description, but will shortly finish the main object, namely, the proof that blast furnace gas can be used in open-hearths, and that there is enough in surplus to run the steel plant when "compound steel" is manufactured.

I refer here to an invention of Mr. Ehrenwerth as long ago as 1883, which relates to the regeneration of blast furnace gas.

In von Ehrenwerth's calculations the absolute calorific value of blast furnace gas of poor quality is given as about 705 Calories per cubic meter, or about 80 B. T. U. per cubic foot.

This gas had a composition of

	Per Cent.
Carbonic oxide	24.23
Carbonic anhydride	20.60
Hydrocarbons	0.38
Hydrogen	0.21
Nitrogen	54.58
	100.00

If this gas were perfectly regenerated it would contain

	Per Cent.
Carbonic oxide	43.42
Carbonic anhydride	0.00
Hydrocarbons	0.21
Hydrogen	0.15
Nitrogen	57.52
	100.00

And its absolute calorific value would be increased to about 120 B. T. U. per cubic foot.

Some time later other authors also made experiments, and

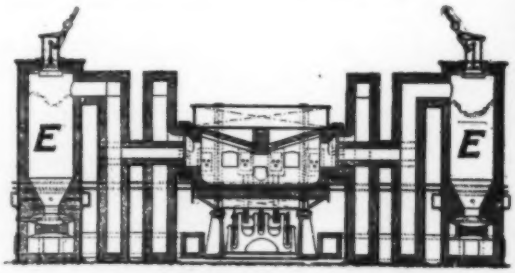


FIG. 3.—VERTICAL SECTION OF COMPOUND FURNACE.

found that blast furnace gas of about 932 Calories would give a suitable furnace heating gas of up to 1,472 Calories, or of a heating value of 160 B. T. U. per cubic foot, better than any standard gas producer gas. Since modern gas engines can also successfully be operated on the *raw* gas and since ore-roasting furnaces, lime-burning cupolas, etc., have also been successfully operated on this raw gas, therefore there is no doubt at all that even high heats as required in melting furnaces can be obtained from blast furnace gas.

In general, an open-hearth furnace of, say, 50 tons capacity, will use about 800 pounds of coal per ton of the product. The output per day is about 100 tons steel, which require, therefore, 80,000 pounds of coal to be gasified in the gas producers in 24 hours, or 3,333 pounds of coal per hour.

One pound of coal yields about 75 cubic feet of producer gas. Hence in our plant we have about 250,000 cubic feet of gas per hour, with an average heating value of approximately 130 B. T. U. One hundred tons of steel require, therefore, 80,000 × 75 × 130 = 780,000,000 B. T. U., or 1 ton, requires 7,800,000 B. T. U. for manufacture.

My intention is now to convert the 576 tons of pig iron in the *same* time in which they are produced into steel of open-hearth quality in a single furnace. If we assume a tap in each 4 hours, 96 tons are tapped six times in a day of 24 hours. Four hours are sufficient to transform this iron into steel, after a preliminary rough fining of the metal by means of the hot blast, besides a modern, say, Talbot continuous process, even if iron ore in lieu of scrap is to be used to a considerable extent. The latter method, however, will turn out profitably only with the high scrap prices of to-day. This all considered,

the 576 tons pig iron will yield in a continuous process about 600 tons of steel in 24 hours, or 25 tons per hour. Each ton requires, as shown before, 7,800,000 B. T. U., so that 195,000,000 B. T. U. per hour are required for 25 tons per hour.

As formerly shown in this article, there are about 175,280,000 B. T. U. for further use left per hour from the blast furnace plant, if we do not take into consideration the improvement of the gas by regeneration. The latter will practically amount to, say, only 20 per cent in heating value; the theoretical result gives 30 and 40 per cent. If we add 20 per cent to above 175,000,000 B. T. U., the available total from the blast furnace gas is increased to 210,000,000 B. T. U.

We have, therefore, a surplus of 15,000,000 B. T. U. over the required 195,000,000 B. T. U. per hour, so that the "regenerators" do not need to be continuously in operation. The coke required to perform this chemical improvement of the gas is easily obtained without cost from the waste of the coke-oven braize and the waste of small coke from the blast furnace plant.

The calculations prove, therefore, that this new process for manufacturing "compound steel" could be worked with blast furnace gas in a rational manner, since the expenditure in gaseous fuel practically falls considerably below that of the equivalent amount of solid fuel.

Any discussion of this subject and any criticism will be very much appreciated by the author of these lines, and any proof of an error in one of the facts or opinions given above will be noted with thanks in the interest of modern progressive work.

DUBOIS, PA.

The Osmotic Pressure before the Faraday Society

At the twenty-sixth ordinary meeting of the Faraday Society, held in London on Jan. 29, 1907, Prof. H. E. Armstrong being in the chair, a general discussion on *osmotic pressure* took place. The discussion was opened by the EARL OF BERKELEY, who exhibited and described his apparatus for the direct measurement of osmotic pressure.

The ordinary direct method of measuring osmotic pressures is to obtain equilibrium on the two sides of the semi-permeable membrane by means of the pressure of a head of liquid. The method devised by the author and Mr. E. G. J. Hartley substitutes mechanical pressure, which is put straight on to the solution and equilibrium thus obtained. Reversing the usual arrangement, the solution is put outside the semi-permeable tube, in an outer containing tube of gun-metal, and the solvent inside, suitable mechanical arrangements being provided for rendering the joints leak-tight. The pressure is obtained from a plunger worked with a pile of weights and delivered through a mercury U-tube to the solution outside the septum. Equilibrium is obtained or calculated by watching through a telescope the motion of the solution in a gauge. The measurements may be made at any desired temperature. Corrections are applied for the imperfect semi-permeability of the membrane by analyzing the solvent to see if any solution has passed into it.

A vapor-pressure method for measuring osmotic pressure was also described. This is a modification of the Ostwald and Walker dynamical method. Air is sucked through a spiral train of glass vessels containing first sulphuric acid, to dry it, then the solution from which it gets saturated up to the vapor pressure of the latter, then the solvent, of which it here takes up more, and finally sulphuric acid again. There are two trains of vessels arranged to oscillate about a central axis, so that the air does not actually bubble *through* the liquid, which is unsatisfactory, but merely passes *over* them, glass beads being placed in the tubes to increase the melting surfaces. The whole apparatus can be placed in a bath and worked at any desired temperature. Arrhennius' relation was used for calculating the osmotic pressures from the lowered vapor pressures, and very concordant results were shown to have been obtained between measurements made by the direct-pressure and the vapor-pressure methods. The author is of opinion that the latter method will chiefly have to be relied upon to furnish accurate data for the further study of osmotics.

Mr. W. C. DAMPIER WHETHAM, F. R. S., speaking on "Indirect Methods of Measuring Osmotic Pressure," agreed as to the importance of the vapor-pressure method. He discussed the formula used by Berkeley and Hartley, and explained the difference between it and the van't Hoff formula obtained from thermodynamic considerations, the expressions being identical where there is no change of volume of the solvent as it enters the solution.

Dr. T. M. LOWRY spoke on "Osmotic Pressure from the Standpoint of the Kinetic Theory." The motion of the particles postulated by the kinetic theory leads in the case of composite gases or liquids to a process of diffusion. When diffusion is resisted a pressure is set up which constitutes the osmotic pressure of the system. This can be effected by means of a semi-permeable membrane, through which only one of the constituents can pass. Such membranes undoubtedly act in virtue of their solvent properties.

The application of the equation $PV = RT$ to the osmotic pressure of gases could be predicted on general theoretical grounds, but there was no *a priori* reason for supposing that it would be applicable to the case of liquids. The mere fact that all liquids were not miscible was sufficient to show that the conditions were widely different from those prevailing in mixed gases. The validity of the law had been established experimentally, and must be accepted as a fact of fundamental importance in all discussions as to the nature of osmotic pressure.

In the early years of the osmotic discussion it had been assumed by van't Hoff and others that since osmotic pressures and gas pressures could be calculated by means of the same formula the conditions must be identical in the two cases, and it was definitely stated that in dilute sugar solutions the osmotic pressure was wholly due to the bombardment of the membrane by the molecules of the sugar, the effects produced by the water molecules being substantially identical on either side of the membrane. The alternative view, that osmotic pressure represented a diminution in the activity or "active mass" of the solvent was suggested by Poynting in 1896, and had subsequently been advocated by Armstrong, Beilby, van Laar and others. In most cases it had been assumed that the essential factor was a diminution in the average energy of the water molecules. From the standpoint of the kinetic theory this was unnecessary, as also was the view advocated by Rodger and by Traube that the solute was directly combined with one or more molecules of the solvent. The mere presence in the surface of the solution of molecules of the solute which could not enter the membrane was sufficient to check the escape of solvent particles from the solution into the membrane, without affecting the reverse passage of solvent from the membrane into the solution. This would lead to a diminished solubility on the side of the solution and to a flow of liquid through the membrane which could only be checked by putting pressure on the solution.

The simplest case of a semi-permeable membrane is undoubtedly to be found in the surface of separation between liquid and vapor. At such a surface the kinetic theory postulates a continual interchange of molecules between the two phases. But while the rate of escape or evaporation would be reduced by the presence of non-volatile molecules in the surface, the rate of condensation would be unaffected, and equilibrium could only be restored by decreasing the vapor pressure and so diminishing the rate of condensation at the surface of the solution. In this case a quantitative relationship could be deduced. If N and n were the relative numbers of particles of solvent and solute in the surface and p and p' the vapor pressures of solvent and solution, then if each molecule of solute replaced a single molecule of solvent in the

surface layer, the reduction of vapor pressure would be shown by the equation $\frac{p'}{p} = \frac{N}{N+n}$. It had been shown by Nernst that from this simple relationship the identity of the osmotic pressure of a dilute solution with the pressure exerted by a gas of equal molecular concentration could be directly deduced. The blocking of the surface was, therefore, sufficient to afford, not merely a qualitative, but also a quantitative explanation of osmotic pressure.

The further discussion of the surface structure of liquids as related to the osmotic phenomena will be printed in the *Transactions*.

Prof. L. Kahlenberg sent in a communication to the discussion on "The Bearing of Actual Osmotic Experiments Upon the Conception of the Nature of Solutions." The occurrence of osmosis and its direction and extent are determined by the nature of the septum and of the liquids that bathe it. Experiments have shown, too, that there is always a major and minor current present, following in opposite directions, although it often appears as if the osmotic process were one-sided. In this case the septum is termed "semi-permeable," and recent research has centered around so-called semi-permeable membranes which really do not exist. The author has demonstrated that it is peculiarly strong selective action on the part of the septum which causes it to be approximately semi-permeable in certain cases, and he has recently even succeeded in separating two colloidal substances by dialysis. This selective action is due to the solubility or insolubility of the substances concerned in the membrane, and therefore osmotic pressure is due to the same forces—essentially chemical in character in the opinion of the author—as the process of solution, and they may be quite variable as different septa and different liquids are employed. The usually accepted "sieve theory" is untenable, because larger molecules frequently go through a membrane more readily than smaller molecules.

So-called semi-permeable membranes have of late years been studied on account of their bearing on van't Hoff's "gaseous pressure" theory of solution and of osmotic pressure. This maintains that as long as the membrane is semi-permeable the osmotic pressure is the same in the case of any septum for a given solution against the pure solvent, a position contrary to experimental facts.

Again, by applying Carnot's cycle and the usual thermodynamic formulæ to the hypothetical case of abstracting, and then returning solvent from a dilute solution in a cylinder fitted with a semi-permeable piston, relations have been deduced between latent heats of fusion and freezing points, and latent heats of vaporization and boiling points, assuming the gas equation to hold for dilute solutions. But the results obtained are frequently far from the experimental values found for different solutes in the case of one and the same solvent (in the case of electrolytes), and it has been necessary to postulate either dissociation or polymerization; but the author, referring to his previous papers, considers these explanations are weak and unnecessary. The thermodynamic calculations involve the assumption of a semi-permeable membrane which at the same time is quite passive. As it is selective action which produces semi-permeability, no such membrane can exist. Therefore, thermodynamic osmotic pressure (which is not really osmotic pressure at all) is quite a different thing from that measured by experiment. Finally, to speak of the experimental osmotic pressure of any solution, without mentioning what septum is used to separate that solution from what other liquid, is devoid of meaning.

Mr. H. N. Morse communicated tables containing a summary of the recent experiments, the results of which are about to be published, made by him and Mr. Frazer, with glucose and cane-sugar. The conclusions arrived at are: (1) In the vicinity of 20° the osmotic pressure exerted by either is equal to that which a molecular equivalent quantity of a gas would exert if its volume were reduced, at the same temperature, to the volume of the solvent in the pure state. (2) Between 18° and 26°, at which the measurements were made, both cane-sugar and glucose in solution are in the anhydrous condition. Measurements made just above 0° yield pressures somewhat above the calculated gas pressures. Measurements with electrolytes are about to be made in which osmotic pressure and dissociation will be determined simultaneously.

Dr. J. C. Philip thought, with reference to Dr. Lowry's hypothesis, that whatever views might be entertained as to the nature of osmotic pressure, one was bound, on thermodynamic grounds, to conclude that dilute solution must behave as a gas in relation to concentration and temperature.

Mr. J. G. A. Rhodin remarked on the fact that osmotic pressure as usually explained seemed to work in the wrong direction, namely, from the lower in the direction of the higher pressure. It was remarkable, too, that a semi-permeable membrane should conduct electricity so easily. Many of the supposed agreements between calculated and observed results and modern physical chemistry were due to a reckless and unjustifiable use of the method of least squares.

Dr. Alexander Findlay said that on account of unfamiliarity with van't Hoff's theory of solutions there had been much misunderstanding as to what was the problem actually to be investigated. Referring to Kahlenberg's objection to semi-permeable membranes, from the thermodynamic standpoint it did not matter whether there were selective absorption or not, provided the changes in energy were equal in opposite directions. He considered that the quantitative side and the thermodynamic side of the subject had been thoroughly worked out, and the remarkable accuracy of the work of Morse and Frazer, and Berkeley and Hartley, pointed to absolute agreement between experiment and the van't Hoff thermodynamic theory. He did not take kindly to Dr. Lowry's kinetic explanations, and thought that the membrane might be left out altogether, and the diffusion only between a solution and a layer of pure solvent on the top of it made use of for explaining what took place.

Mr. W. A. Davis said that van't Hoff's original hypothesis neglected the influence exerted on the solute by the solvent; Dr. Lowry's modification of van't Hoff's conception was therefore necessary from that standpoint. Osmotic pressure had also been explained as the tendency to equalize the surface tension on the two sides of the osmotic cell.

The Earl of Berkeley disagreed with Dr. Findlay that the thermodynamic was the only logical way of looking at things. He discussed further the discrepancy between the formula used by him and the thermodynamic formula which could be explained on the conception one formed of osmotic pressure.

Dr. T. M. Lowry said that the views he advocated accounted for the negative direction of the osmotic flow referred to by Mr. Rhodin. It was not remarkable that Boyle's law should hold good for osmotic pressure, but it was remarkable that the constant R in the two cases should be identical. The agreement should throw light on the surface structure of liquids.

Prof. H. E. Armstrong, the chairman, emphasized the general applicability of the vapor-pressure method, which, although tedious and troublesome, could be used with any solvent, at all temperatures, and under all pressures. The compressibility of liquids would prevent it agreeing quite with direct measurements; it was therefore important that careful comparisons of the two should be made. In reply to Dr. Findlay, Morse and Frazer's results did not agree with those of Berkeley and Hartley; there was a possible constant error in the former, in spite of the extraordinary accuracy of the work. He hoped more experimental work would be done with living tissues on the line of Pfeffer's original measurements. The term "osmotic pressure," unless merely used in the sense of a pressure required to establish equilibrium, was a misleading one; there was nothing to show that a pressure existed inside a solution in a closed vessel apart from simple gravity pressure. The

results of Morse and Frazer brought dilute and concentrated solution into line, and were a great advance on the van't Hoff treatment; they put the ordinary gas point of view quite out of court. He depreciated the dogmatic, unscientific treatment to which the whole subject had been subjected for the purpose of bolstering up a particular point of view: the facts needed examination from a broad and critical standpoint.

The authors reserved their full replies for the printed discussion in the *Transactions*.

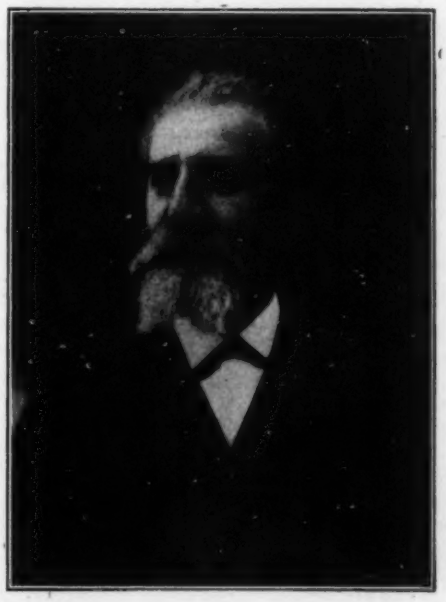
Professor Henri Moissan.

Two Obituaries.

I.

Science has been cruelly robbed of one of her most brilliant and indefatigable workers, in the death of M. Henri Moissan, which occurred at Paris, on Wednesday, Feb. 20, after an operation for appendicitis.

Although Henri Moissan has for long been known throughout the whole world as one of the foremost chemists of the age, many have misunderstood the nature of the claims upon which his fame is founded. It may, therefore, be of some interest to emphasize certain leading traits of his character and recall some of the numerous brilliant researches which have been carried out in his laboratory.



Henri Moissan

In the first place, it is necessary to remember that Moissan's early work as a student was carried out under the guidance of Henri Sainte Claire Deville and Debray, and that his whole life work worthily upholds the traditions and high quality of that brilliant French School of "Chimie Minérale," led by Deville, and which has, throughout, been noteworthy for genius and resource in experimentation.

Henri Moissan was born on Sept. 28, 1852, in Paris, and at an early age developed an interest in chemistry.

His first research, published in 1877, on the oxides of the iron group was amplified and presented in 1880 as a thesis for the doctorate of science in the "Faculté des Sciences" at Paris.

Up to this time Moissan held various minor posts at the Natural History Museum and elsewhere, but most of his energy seems to have been devoted to his research work. In 1886, soon after his isolation of fluorine, he was elected professor of toxicology in the "école superieure de Pharmacie," as it was desired to retain him in Paris and this was the only post of a chemical nature available at that time. Only in 1899 did Moissan give his first course of chemistry lectures, being appointed professor of inorganic chemistry at the same institution. Finally, in 1901, he was elected professor at the Sorbonne to fill the most important chair of chemistry in Paris. In 1891 he was elected a member of the Academy of Sciences, the crowning honor of a French scientific man.

Although a Parisian by birth and residing all his life in this, the gayest capital of Europe, Moissan never took any considerable part in social or political affairs. He was of a retiring disposition and was intensely devoted to his science; he has himself clearly expressed the determined way in which he cultivated the "simple life": "Ma vie a eu toute la simplicité de ma carrière de professeur et mon existence s'est partagé heureuse jusqu'ici entre mon laboratoire et ma maison."

Even during the long tenure of his professorship of toxicology, Moissan maintained a research laboratory and attracted to it a number of chemists, some even from distant countries. His pupils have one and all carried away an affectionate remembrance of his interest and encouragement in their work. As a lecturer, particularly in his public discourses and in the excellent courses on inorganic chemistry which he gave in the last few years of his life, Moissan was distinguished even amongst French chemists by his brilliant oratory and his skill in experimental demonstration.

As a scientific worker, Moissan's fame rests on two great researches: in the first, he overcame the difficulties which had baffled the experimental skill of Sir Humphrey Davy, Faraday, Frémy and a host of other workers, and at last isolated the element fluorine; in the second, he first showed how the electric arc might be utilized as a valuable instrument for chemical research and prepared many hitherto rare and refractory elements and several series of new compounds of great interest in chemical science.

On June 28, 1886, Moissan communicated to the Academy of Sciences the first details of his work on the preparation of fluorine by the electrolysis of anhydrous hydrofluoric acid, and in the following month gave a more complete account, which enabled the importance of his work to be realized. The success thus achieved was in no way due to chance; the record of the work indicated the logical trial, abandonment and improvement of a number of other likely methods of preparation. The difficulties were well known to all chemists, since for more than seventy-five years memorable attempts had been made to isolate this, the last of the unliberated elements. That the work was not without danger may also be gathered from the fact that at least two professors of chemistry had succumbed to the poisonous and corrosive effect of the vapors of anhydrous hydrofluoric acid. His reputation as an experimental chemist of the first rank was soundly established so soon as the truth of his results were proved. But for some years other experimenters, not succeeding in repeating Moissan's work, questioned the accuracy of his results. In 1893 the question had been so debated in England that Moissan sent over his chief assistant, M. Meslans, to demonstrate the isolation of fluorine before the British Association at Nottingham.

The main reason which impelled him to pass from the fluorine investigations to the study of chemical reactions occurring at high temperatures seems to be closely connected with a desire, which he had long entertained, to solve the mystery of the origin of the diamond. The hope that the great affinity of fluorine for other elements would help him in his quest was not realized. He was, however, led to a methodical study of the behavior and transformation of the three allotropic modifications of the element carbon-diamond, graphite and charcoal. From a study of the diamond as it occurs in nature he came to the conclusion that immense pressure was the determining factor in its production. A proof of this seemed to be supplied to him by a meteorite from the Diablo Canon, which, in the middle of a mass of iron, revealed two small transparent diamonds, surrounded by amorphous carbon in compressed strips. "Nature," he says, "seems to have been caught in the act."

How he planned and successfully carried through the adaptation of this idea in the laboratory with the successful production of minute diamonds is well known to all.

It was for the purpose of augmenting the quantity of carbon dissolved in iron that Moissan first required and adopted the electric furnace.

In his electric furnace work Moissan's pre-eminent position is due, not to the design or discovery of a special form of furnace, but rather to the skill with which he investigated in detail a number of individual chemical reactions. In each case he devoted great care to the purification and analysis of the raw materials required in the process, and submitted the products to careful examination and quantitatively determined their composition. Thus his preparation of chromium, tungsten, molybdenum, uranium, vanadium and many other metals in a fused and coherent form greatly enriched our knowledge of the chemical and physical properties of these elements.

Of still greater importance was the methodical following up of the chance formation of calcium carbide which he observed around the carbon electrodes in his early furnace experiments. From this observation he was led to discover, and fully determine the nature and properties of, a large number of metallic carbides, borides and silicides, many of them hitherto absolutely unknown, or like the metals mentioned above, only obtainable in impure and fragmentary specimens handed down from one professor of chemistry to another as an heirloom of priceless value. There is, perhaps, no need to consider at the present time in how far industry is directly indebted to Moissan's work. He himself had invariably expressed his desire not to be considered in such discussions, and so far as the merit of his work is concerned it needs no support of this nature. Indirectly both industry and science have benefited enormously.

On the Continent of Europe his scientific investigations are directly credited with a renaissance in the field of inorganic chemistry, which, particularly in Germany, had been almost entirely neglected for the more remunerative field of organic chemical research. Even in England and America where the study of inorganic chemistry has always been pursued with activity, this work has been of no little assistance in instilling enthusiasm and encouraging the deeper study of such problems.

In recent years the scientific work which Moissan had carried out was still directed along the two main lines of his activity; by consolidating and extending his earlier work upon the properties of fluorine and its compounds, on the one hand, and the various physical and chemical phenomena associated with the intensely high temperature of the electric arc on the other, he has largely increased the debt which science owes to his labors. Henri Moissan is deeply mourned by many friends and admirers, who, only in December last, welcomed the well-merited bestowal of the Nobel prize, in recognition of his work, and themselves were just on the point of presenting him with a medal to commemorate the twentieth anniversary of his work on fluorine.

May his memory long remain with us and in the hands of his many admirers, lead to the continuity of the work to which he was so greatly devoted. R. S. Hutton.

Manchester, England.

* * *

II.*

"But what I cannot put into these successive chapters is the joy I experienced in pursuing these discoveries. To walk in a newly turned furrow, to feel one's self untrammeled, to see new subjects for study spring up on all sides, is so delightful that it cannot be fully comprehended except by those who have experienced the ardent pleasure of research."

With this closing to a preface, Moissan throws open the door and invites us to participate with him in the discoveries made with the electric furnace. This most skillful of experimenters, having won his spurs in the conflict of isolating fluorine, thus distancing all others who had attempted the task, he attacks new problems with zeal and the ability of an accomplished craftsman. He was truly the embodiment of Davy's adage, "It is the duty of the chemist to be bold in pursuit."

Almost his last words as he left this city in 1904 was the terse sentence, "Chemistry is still an experimental science."

Moissan's indefatigable pursuit of a problem is typically exemplified in a remark in his monograph on "Fluorine and its Compounds": "And so after three years of research I got to the first important experiment in the isolation of fluorine."

Another instance of his indomitable persistence will be remembered by some who were fortunate enough to hear his brilliant lecture, delivered at the College of Physicians and Surgeons, in 1896, when, at the invitation of the New York Academy of Sciences, the American Chemical Society, the American Institute of Electrical Engineers, and the College of Pharmacy, he demonstrated the use of his electric furnace, and showed his method of making diamonds. As he removed the glowing crucible from within the furnace and plunged it into a mass of water, some of his audience, all scientists, involuntarily moved as if expecting an explosion. "Have no fear," said he, "I have performed this experiment without accident over three hundred times."

In friendly appreciation of the honors done him on that visit to the United States, he sent to the National Museum a collection of sixteen specimens of the products he had obtained by the use of this new instrument of research. The bottles were neatly closed with parchment, and on each he had placed his signature. This collection will now be a unique record of his genius. During the same visit Moissan lectured at the Chicago University, and attended the Sesqui-Centennial Celebration at Princeton. He made a study of our educational institutions, and on his return wrote a report, in which he dilated on the advantages our country was deriving from the liberality of her citizens towards the institutions of learning, and he exerted all his influence, which constantly increased, to bring about the same state of affairs in France, where, because of the University belonging to the State, endowments were rarely forthcoming.

In 1889 he published some new researches on the "Isolation of Fluorine," and in 1890 he wrote on the "Fluoride of Carbon." He received the "Prix Lacaze" in 1887.

His work on the fluoride of carbon led to his undertaking a study of all three forms of carbon, amorphous, graphite and diamond, and it was the object of subjecting carbon to excessively high temperatures that he devised his electric furnace, a simple yet all powerful instrument. With it he undertook a long series of researches on the elements and some of their compounds, the story of which would not only take us through many a chapter of science but also unfold the beginnings of a new art in chemistry. Later, together with Dewar, he liquefied fluorine, and showed how it would combine spontaneously with hydrogen, thus establishing, at least so far, the lowest temperature at which chemical union takes place.

With this electric furnace he has demonstrated not only the indestructibility of many of the elements by the highest heat yet attained, for in a brilliant discourse given at Rome in 1906, at the International Congress of Applied Chemistry, he detailed his distillation of gold, the platinum metals, chromium, iron, nickel, manganese, tungsten, molybdenum and uranium, and before he had also demonstrated in the preparation of titanium carbide and other compounds that chemical combination took place at the highest temperatures at our command. What a technique! It is as if some master of the piano began at the bass notes and swept up majestically to the highest treble.

Moissan studied particularly the chemistry of the elements. His isolation of calcium is a further example of new methods of attacking his favorite problem. At the International Con-

* Address before the New York section of the American Chemical Society.

gress at Berlin he gave a lecture which charmed his audience and elicited the highest admiration. He presented his results on the hydrides of the alkalies and the alkali-earth metals. He added with these a further demonstration that hydrogen is not a metal, by showing that potassium and sodium hydrides, KH and NaH, do not conduct electricity, and are therefore not alloys.

We find him passing readily from the realm of inorganic to that of organic chemistry, producing potassium formiate, HCOOK, by the direct union of KH and CO_2. One can never read many pages of his memoirs without being struck with the fact that he made no sharp lines of demarcation between the inorganic and the organic; to him they were but one chemistry.

With a host of collaborators, Moissan was engaged to the last on a great work—"Chimie Minerale." A discourse given at the Convocation of Scientists at the St. Louis Exhibition in 1904, delivered at the invitation of the Government, was a resumé of his introduction of this important treatise.

During the last few years Moissan had forged ahead. He was the president of the International Congress of Applied Chemistry in Paris in 1900. He had previously been appointed to Friedel's Professorship at the Sorbonne, and with highest inspiration he was devoting his irrepressible energies to many problems. These were, indeed, multitudinous. His strictly scientific work was always paralleled with important applications for the well being of the human race. His originality showed him as a leader of thought and also a worker for his fellow beings, inaugurating new industries, planning new manufactures. Few chemists have had wider influence. A medal struck in honor of the twentieth anniversary of the isolation of fluorine was only recently given by his students and friends.

The recognition of all this was freely given to him, not only at home, for France is loyal to her sons, but also abroad. The last highest honor conferred on him was the Noble prize for chemistry in 1906.

But to those who came within the subtle influence of his personality the fact that an operation for appendicitis, which resulted fatally on Feb. 20, 1907, has severed a friendship with one whose charm of manner endeared him to students, associates and friends alike, has come as a blow so sudden and unexpected that it is difficult to fully realize it. Our sympathy goes out to his noble wife, that faithful amanuensis, who has indefatigably taken down the thoughts of the adored husband and preserved them for us, and to the young man, his only child, who is devoting himself to the same profession.

New York City. Charles A. Doremus.

Iron and Steel Statistics.

The following statistics of iron and steel production in 1906 have been returned:

United States.

The regular official statistics of the American Iron and Steel Association show pig iron at 25,307,191 gross tons, a fresh record, supplanting that made in 1905, 22,992,380 gross tons. Bessemer steel ingots and castings, 12,275,253 gross tons, against the record of 10,941,375 gross tons in 1905. Bessemer steel rails, by the makers of Bessemer steel ingots, 3,705,642 gross tons, against 3,135,729 gross tons in 1905, both records. The complete rail statistics will include the small tonnage of Bessemer steel rails (chiefly light rails) made by mills which do not produce Bessemer steel ingots, also iron rails and open-hearth steel rails.

Canada.

Pig iron, 541,957 gross tons, against 468,003 gross tons in 1905; both records; authority, American Iron and Steel Association.

Great Britain.

Pig iron, 10,149,388 gross tons, against 9,592,737 gross tons in 1905; both records; authority, British Iron Trade Association; the statistics of the home office appear later; as a rule, the two sets of statistics agree closely and are equally authoritative.

Germany.

The statistical department of the German Association of Iron and Steel Producers reports the production of pig iron at 12,478,068 metric tons (equal to 12,280,877 gross tons), against 10,987,623 metric tons in 1905; both records.

United States Steel Corporation.

Blast furnace products, including pig iron, ferromanganese, spiegeleisen and ferrosilicon, 11,267,377 gross tons, against 10,172,148 gross tons in 1905. Bessemer and open-hearth steel ingots, 13,511,149 gross tons, against 11,995,239 gross tons in 1905. All steel products for sale, 10,578,433 gross tons, against 9,226,386 gross tons in 1905.

Lake Superior Iron Ore Shipments.

Lake Superior iron ore shipments, 38,522,139 gross tons, against 34,353,456 gross tons in 1905, both records.

World's Pig Iron Production.

The statistics of pig iron production in the United States, Canada, Great Britain and Germany, already noted, make a total of 48,279,413 gross tons, against 43,867,099 gross tons by the same countries in 1905. Allowing an increase of about a million tons by the other producing countries would indicate a grand total for the world in 1906 of about 59,000,000 gross tons.

Metallurgical Calculations.

By J. W. Richards, Ph. D.

Professor of Metallurgy in Lehigh University.

Problem 70.

In the open-hearth furnace of Problem 69 (this journal, March, 1907, p. 81) the calculations therein made showed that, per heat of steel produced, 5,191 kilograms, there entered and left the furnace the following volumes of gases, measured at standard conditions:

	Producer Gas.	*Air.*	*Chimney Gases.*
CO^2	235 m^3	....	1,830 m^3
O^2	60 m^3	3,833 m^3	1,116 m^3
CO	1,467 m^3	....	....
CH^4	26 m^3	....	....
H^2	539 m^3	....	....
N^2	3,833 m^3	9,679 m^3	13,512 m^3
H^2O	726 m^3	211 m^3	1,500 m^3

The temperature of air used was 26°, of producer gas 165°, of chimney gases 400°. The excess of air used was 48.5 per cent. The gas and air entered the laboratory of the furnace preheated to 1,100°, and the products of combustion entered the regenerators at 1,450°. Items of heat balance sheet:

Available.

	Calories.
In warm charges	189,210
Sensible heat of air used at 26°	90,480
Sensible heat of gas used at 165°	360,550
Heat of combustion of the gas	6,202,300
Heat of oxidation of the bath	833,600
Heat of formation of slag	24,200
	7,700,340

Distribution.

In melted steel at tapping at 1,410°	1,437,900
In slag at tapping at 1,410°	238,000
Decomposition of limestone	9,200
Sensible heat in chimney gases at 400°	3,065,350

All other losses, not classified..... 2,949,890

7,700,340

Required:

(1) The thermal efficiency of the regenerators.
(2) The thermal efficiency of the laboratory of the furnace.
(3) The temperature of the flame.
(4) The change in (3) if only the theoretical amount of air for combustion were used.

Solution:

(1) The products of combustion, entering the regenerators at 1,450°, carry into them the following amounts of heat:

		Calories.
CO^2	$1,830 \times 0.689 =$	1,261
$O^2 + N^2$	$14,628 \times 0.342 =$	5,003
H^2O	$1,500 \times 0.557 =$	836
Calorific capacity per 1°		= 7,100

Calorific capacity per 1,450° = 10,295,000 Calories.

The same gases entering the chimney flue at 400° carry with them the following amounts:

		Calories.
CO^2	$1,830 \times 0.458 =$	838.1
$O^2 + N^2$	$14,628 \times 0.314 =$	4,593.2
H^2O	$1,500 \times 0.400 =$	600.0
Calorific capacity per 1°		= 6,031.3

Calorific capacity per 400° = 2,412,500 Calories.

The gas used for combustion, entering the regenerators at 165° and leaving it at 1,100°, carried into the regenerators 360,550 Calories, as already given in the balance sheet, and carried out at 1,100° the following:

		Calories.
CO^2	$235 \times 0.612 =$	143.8
CH^4	$26 \times 0.620 =$	....
O^2, N^2, H^2, CO	$5,899 \times 0.333 =$	1,964.4
H^2O	$726 \times 0.505 =$	366.6
Calorific capacity per 1°		= 2,474.3

Calorific capacity per 1,100° = 2,722,300 Calories.

Heat abstracted from regenerators:

2,722,300 — 360,550 = 2,361,750 Calories.

The air used, entering the regenerators at 26°, carries in as sensible heat 90,480 Calories, as already given in the balance sheet, and issuing from them at 1,100° carries out the following:

		Calories.
$O^2 + N^2$	$13,512 \times 0.333 =$	4,499.5
H^2O	$211 \times 0.405 =$	85.5
Calorific capacity per 1°		= 4,585.0

Calorific capacity per 1,100° = 5,043,500 Calories.

Heat abstracted from regenerators:

5,043,500 — 90,500 = 4,953,000 Calories.

The thermal efficiency of the regenerators may now be calculated from three standpoints. There is no doubt that the gas and air take from the regenerators, and return to the body of the furnace 2,361,750 + 4,953,000 = 7,314,750 Calories. This is, therefore, the usefully-returned heat, and the ratio of this to the heat received by the regenerators measures their efficiency *qua* regenerators. The three figures obtained for this efficiency depend on what is to be considered as the heat chargeable against the regenerators. Are they to be charged with all the heat in the hot products at 1,450° (10,295,000 Calories), or only with the heat left in the regenerators by these products, leaving at 400° (10,295,00 — 2,412,500 = 7,882,500 Calories), or perhaps only with the heat carried in less a certain assumed amount representing the minimum temperature to which it is desirable to cool the products before they enter the chimney?

If we charge the regenerators with all the heat *brought in* by the products their thermal efficiency figures out:

$$\frac{7,314,750}{10,295,000} = 0.71 = 71 \text{ per cent.} \qquad (1)$$

If we charge them with the heat *left in* the regenerators, by the products, their efficiency is:

$$\frac{7,314,750}{7,882,500} = 0.93 = 93 \text{ per cent.} \qquad (1)$$

leaving 7 per cent of the heat chargeable against them lost by radiation from their walls and conduction to the ground.

If we think that the first calculation gives too low an efficiency, because the gases must leave the regenerators hot, in order to be used for chimney draft, and therefore some or all of the heat in the chimney gases should not be charged against the regenerators, on the other hand, the second calculation may represent too high an efficiency, because the gases may be discarded to the chimney at a higher temperature than is necessary to provide the requisite chimney draft, and this *excess* of chimney temperature and consequent heat loss is a defect of the regenerators which they should be charged with. If we assume that a chimney gives very nearly its maximum drawing capacity with the gases entering it at 300°, it would be perfectly proper to charge the regenerators with all the heat which could be given out by the products in cooling from 1,450° to 300°, heat which they should have entirely absorbed. In the case in hand, the products at 300° would contain (by calculations similar to those already made) 1,777,400 Calories, leaving 10,295,000 — 1,777,400 = 8,517, 600 Calories chargeable against the regenerators, as the heat which they should have absorbed or intercepted. Measured by this standard, their efficiency is:

$$\frac{7,314,750}{8,517,600} = 0.86 = 86 \text{ per cent.} \qquad (1)$$

The losses from the regenerators in this view would be 7 per cent (about) by radiation and 7 per cent by *unnecessary* chimney loss.

Comparing these three methods of considering efficiency, the third appears to the writer as the fairest, and the one which gives the metallurgist the most reliable criterion of the real work which his regenerators are doing for him, and the best basis of comparison when considering the work of different regenerators or of the same regenerators under different conditions.

(2) The laboratory of the furnace, the space enclosed between the hearth, roof, side walls and ports, receives from the entering preheated gas and air their sensible heat, and the heat generated by combustion. In the case in point these items total, as already calculated:

	Calories.
Sensible heat in preheated gas.....	2,722,300
Sensible heat in preheated air.....	5,043,500
Heat generated by the combustion.....	6,202,300
Total	13,968,100
Hot products at 1,450° take out.....	10,295,000
Heat left in the laboratory.....	3,673,100

These figures show that the laboratory of the furnace appropriates to its own purposes 3,673,100 Calories out of the 13,968,100 Calories poured into it. This part of the furnace, therefore, *qua* laboratory, has an efficiency in this respect of

$$\frac{3,673,100}{13,968,100} = 0.26 = 26 \text{ per cent.} \qquad (2)$$

This is the datum which would be useful to the metallurgist in comparing the efficiencies of differently shaped laboratories,

such as those with differently shaped hearths, differently shaped roof, differently arranged ports, etc. This conception of efficiency is that taken by Damour, and repeated by Queneau in his book on "Industrial Furnaces." We must be careful here not to compare the heat left in the laboratory with the heat of combustion alone. This would give

$$\frac{3{,}673{,}100}{6{,}202{,}300} = 0.59 = 59 \text{ per cent.}$$

But this is not the heat absorption of the laboratory alone, but is a function of the furnace as a whole, and depends largely on the efficiency of the regeneration accomplished by the regenerators. If we wish to obtain an idea of the perfection with which the laboratory of the furnace appropriates the heat passing into and through it, we must simply compare what it appropriates with what was sent into it. The percentage of this appropriation measures the efficiency of the laboratory for abstracting heat for the purpose of heating itself—a datum highly useful to know if the furnace is used simply for keeping a given working space up to a given temperature for an indefinite time. If the heating of the furnace charge to the furnace heat is only a minor part of the useful work of the furnace, and keeping it at that temperature is the chief function of the furnace, then the relation of the heat thus appropriated and utilized to the total heat available to the laboratory, is a measure of the perfection of construction of the laboratory. This is the view of Damour and Queneau, and seems in some respects plausible.

However, if we are examining this question thus in detail, it appears to the writer that the view just explained and the conclusions derived therefrom may really be the very opposite of the truth of the matter, and lead to entirely erroneous conclusions. If two exactly similar open-hearth furnaces are constructed, with the sole difference that the body of one is thicker walled than the body of the other, it is perfectly true that maintaining the same temperature in each will require more gas in the thin-walled furnace, but that it will abstract and radiate a considerably larger proportion of the heat passing through it than the thick-walled furnace. Yet, according to Damour and Queneau's principle, the thin-walled furnace would be considered as being the more efficient laboratory of the two. The truth is, we must either compute the heat left in the laboratories per unit of time per cubic meter of stock space in order to compare different furnaces, or else to get absolute efficiency compute the ratio of the heat passing into the laboratory with that absorbed usefully by the charge, viz.: in this case:

$$\frac{1{,}447{,}100}{13{,}968{,}100} = 0.104 = 10.4 \text{ per cent.} \qquad (2)$$

(3) The temperature of the flame is that to which the products of combustion can be raised by the maximum heat which they contain. When perfectly consumed they contain the heat pre-existing as sensible heat in the hot gas and air, plus the heat of combustion. We have already calculated this total as 13,968,100 Calories. The mean heat capacity of the products of combustion, per 1° between 0° and t°, is:

CO^2	1,830 m³ (0.37 + 0.00022t)	= 677 + 0.4026t
$O^2 + N^2$	14,628 m³ (0.303 + 0.000027t)	= 4,428 + 0.3950t
H^2O	1,500 m³ (0.34 + 0.00015t)	= 510 + 0.2250t
	Sum	= 5,615 + 1.0226t

Therefore,

$$t = \frac{13{,}968{,}100}{5{,}615 + 1.0226t}$$

Whence

$$t = 1{,}749^\circ \qquad (3)$$

(4) If the excess air were not used, the 1,116 cubic meters of oxygen in the products, together with its corresponding quantity of nitrogen, 4.249 cubic meters, would be absent from the products of combustion, as would also some of the 211 cubic meters of water vapor. Since the oxygen in the air was 3,833 m³ and the unused excess 1,116, the proportion of the 211 m³ of water vapor belonging to the excess air was

$$211 \times \frac{1{,}116}{3{,}833} = 61 \text{ m}^3,$$

and the final products are CO^2 1,830 m³, N^2 9,263 m³ and H^2O 1,439 m³.

The heat available will be the same as before from combustion, the same from preheated gas, but less from preheated air, because of the absence of the excess air. Since the air formerly used brought in 5,043,500, the proportion of this carried by the excess air is

$$5{,}043{,}500 \times \frac{1{,}116}{3{,}833} = 1{,}468{,}500,$$

leaving heat brought in by air in the second case the difference, viz.: 3,575,000, and the total heat in the products 13,968,100 — 1,468,500 = 12,499,600 Calories.

The mean heat capacity of the diminished quantity of products, per 1°, is, by the same methods as previously used, 3.973 + 0.8685t, and, therefore,

$$t = \frac{12{,}499{,}600}{3.973 + 0.8685t} = 2{,}142^\circ \qquad (4)$$

a gain of nearly 400° in maximum temperature, by cutting off the 40 per cent excess of air which was being used.

Problem 71.

The new style Siemens furnace for small plants has the gas producers built in as part of the furnace, between the two regenerators. The furnace has only one regenerative chamber at each end for preheating the air used for burning the gas, while the latter comes to the ports hot directly from the producers. This plant is compact, economical of fuel, allows high temperatures to be reached, and occupies little floor space. As compared with the old style separate furnace and producer plant, it occupies about half the floor space, costs about 60 per cent as much to build and uses about 60 per cent as much coal. They are largely used abroad for melting steel for castings in small foundries and for melting pig iron for castings to be subsequently annealed to malleable castings.

Such a furnace melted a charge of 3,000 kilograms of cast iron in 4 hours, using 750 kilograms of coal. The cast iron was charged cold, at 0°, and run out melted at 1,450°, containing 300 Calories of sensible heat per kilogram. The coal and gases were of the following composition:

Coal—C 75, H 5, O 12, H^2O 2, ash 6.
Producer Gas—CO 20, CO^2 5, CH^4 2, H^2 16, N^2 57.
Chimney Gas—CO^2 19, O^2 1.8, N^2 79.2.

The air coming to the furnace is at 0° and dry; a steam blower is used to run the producer, using 1 kilo. of steam per 6 kilos. of air blown in, and the raising of this steam requires 0.1 kilo. of coal used under boilers. The ashes produced weigh 75 kilos. per heat run. Temperature of the hot producer gas entering the furnace laboratory 600°, of preheated air 1,000°, of products leaving laboratory 1,400°, of products entering chimney flue 350°.

Required:

(1) A heat balance sheet of the furnace as a whole.
(2) The net thermal efficiency of the furnace.
(3) The net thermal efficiency of the laboratory of the furnace.
(4) The net thermal efficiency of the regenerators.
(5) The theoretical flame temperature in the furnace.

Solution:

(1) Heat balance sheet per charge of 3,000 kilos.

Heat Available.

	Calories.
Calorific power of coal used in producers....	5,248,500
Calorific power of coal used under boilers...	398,900
	5,647,400

Heat Distribution.

In melted cast iron.....	900,000
In chimney products.....	768,950
Loss by unburnt carbon in ashes.....	243,000
Used for raising steam.....	398,900
Loss by radiation and conduction.....	3,336,550
	5,647,400

Calculation of the Heat Balance.

The heat of combustion of a kilogram of coal is not given, therefore must be calculated from its composition:

		Calories.
C $0.75 \times 8{,}000$	=	6,075
H $\left(0.05 - \frac{0.12}{8}\right) \times 34{,}500$	=	1,208
Calorific power to liquid water	=	7,283
Vaporization heat of water formed $(0.02 + 0.45) \times 606.5$	=	285
Calorific power to vapor of water	=	6,998
Of 750 kilos. in the producers	=	5,248,500

The coal used under the boilers can only be found by first finding how much steam was used, which in its turn can be gotten from the air blown in, and the nitrogen of this can be found from the total nitrogen in the producer gas. The volume of producer gas can be gotten from its carbon content per cubic meter and the known weight of carbon gasified. Or, turning this chain of reasoning the other way, if we subtract the carbon in the ashes from the carbon in the coal, the difference is the carbon entering the gases; this divided by the carbon in 1 cubic meter of chimney gas (calculated from its analysis) gives the volume of chimney gas, and by the carbon in 1 cubic meter of producer gas gives the volume of this gas used, from which the nitrogen in it can be calculated, which divided by 0.792 gives the volume of air used, from which its weight is obtained, thence the amount of steam used, and finally from this the amount of coal necessary to burn under boilers to raise this steam.

[In working most metallurgical problems the difficulty of finding the connection or succession of steps connecting the requirements with the data given is often easiest overcome by starting with the requirement in mind and noting from what other figure its value may be calculated, and thus passing backwards from one figure to another we finally arrive at one which can be found directly from the data given. The logical sequence of operations is thus disclosed and the problem is in reality solved; the following of the thread in the reverse direction, from data to requirement, involves calculations only, not hard thinking, and is usually a matter of the simplest arithmetic.]

The operations for calculating the steam used are as follows:

Carbon in coal in producers $= 750 \times 0.75$	=	562.5 kg.
Carbon in ashes from producers $= 75 - 45$	=	30.0 "
Carbon in gases from producers	=	532.5 "
Carbon in producer gas per 1 $m^3 = 0.27 \times 0.54$	=	0.1458 "
Producer gas per heat $= 532.5 \div 0.1458$	=	3,652 m^3
Nitrogen in producer gas $= 3{,}652 \times 0.57$	=	2,082 "
Air used in the producer $= 2{,}082 \div 0.792$	=	2,629 "
	=	3,399 kg.
Carbon in chimney gas per 1 $m^3 = 0.19 \times 0.54$	=	0.1026 "
Chimney gas per heat $= 532.5 \div 0.1026$	=	5,190 m^3
Nitrogen in chimney gas $= 5{,}190 \times 0.792$	=	4,110 m^3
Nitrogen from air used $= 4{,}110 - 2{,}082$	=	2,028 "
Air used to burn gas $= 2{,}028 \div 0.792$	=	2,561 "
Weight of this air $= 2{,}561 \times 1.293$	=	3,311 kg.
Weight of air used in producer	=	3,399 "
Weight of steam used in producer $= 3{,}399 \div 6$	=	567 "
Weight of boiler coal used $= 567 \times 0.1$	=	57 "
Calorific power of this coal $= 6{,}998 \times 57$	=	398,900 Cal.

The heat in the melted cast iron is simply its weight times $300 = 300 \times 3{,}000 = 900{,}000$ Calories.

(2) The net thermal efficiency follows directly from this, as

$$\frac{900{,}000}{5{,}647{,}400} = 0.16 = 16 \text{ per cent.} \qquad (2)$$

(3) The gaseous products consist dry, as analysed, of

CO^2	$5{,}190 \times 0.19$ =	986 m^3
O^2	$5{,}190 \times 0.018$ =	93 "
N^2	$5{,}190 \times 0.792$ =	4,110 "

And in addition all the water formed from the coal, plus the steam used, a total of:

Water from coal $= 0.47 \times 750$	= 352.5 kg.
Steam	= 566.5 "
Total water vapor in gases	= 919.0 "

Volume at standard conditions $= 919 \div 0.81 = 1{,}135$ m^3

At 350° these products carry out of the furnace heat as follows:

		Calories.
$O^2 + N^2$	$4{,}203 \times 0.312$ =	1,311
CO^2	986×0.447 =	441
H^2O	$1{,}135 \times 0.392$ =	445
Average caloric capacity per 1° =		2,197
Capacity per 350° = 768,950 Calories.		

These same products would carry heat out of the laboratory of the furnace at 1,400° as follows:

		Calories.
$O^2 + N^2$	$4{,}203 \times 0.341$ =	1,433
CO^2	986×0.678 =	669
H^2O	$1{,}135 \times 0.550$ =	624
Average caloric capacity per 1° =		2,726
Capacity per 1,400° = 3,816,400 Calories.		

The heat developed in the laboratory of the furnace by the combustion of the 3,652 m^3 of producer gas is

		Calories.
CO	$0.20 \times 3{,}062$ =	612
CH^4	$0.02 \times 8{,}598$ =	172
H^2	$0.16 \times 2{,}613$ =	418
Per 1 m^3 =		1,202
Per 3,652 m^3 = 4,389,700 Calories.		

The 3,652 m^3 of gas comes into the laboratory of the furnace at 600°, and therefore carries in as sensible heat the following:

CO^2	$0.05 \times 0.502 = 0.0251$	Cal. per 1°
CH^4	$0.02 \times 0.512 = 0.0102$	" "
CO, N^2, H^2	$0.93 \times 0.319 = 0.2967$	" "
	$0.3320 \times 600 = 199.2$ Cal	

Per 3,652 $m^3 = 199.2 \times 3{,}652 = 727{,}480$ Calories.

This does not count, however, the water vapor accompanying the producer gas. This is best determined from the fact that the total hydrogen in the coal and steam used in the producer must appear as hydrogen in the producer gas in some form or other, and whatever is not present in the dry producer gas must be in the water vapor accompanying it. The amount of this and the heat it carries into the laboratory of the furnace are thus determined:

Hydrogen in 750 kg. coal = 750 × 0.0522 = 39.15 kg.
Hydrogen in steam used = 567 ÷ 9 = 63.00 "
Hydrogen going into producer gases = 102.15 "
Hydrogen in dry producer gases,
3,652 × 0.20 × 0.09 = 65.74 "
Hydrogen in producer gas as vapor of
water = 36.41 "
Water vapor in producer gas = 327.7 "
Volume of this vapor = 405 m^3
Heat in this at 600°, 405 × 0.43 × 600 = 104,500 Cal.
Total heat in moist producer gas,
727,480 + 104,500 = 831,980 "

We must also calculate the heat brought into the laboratory of the furnace by the preheated air, at 1,000°, used for combustion. This is

2,561 m^3 × 0.330 × 1,000 = 845,130 Calories.

It follows from the above calculations that the laboratory of the furnace receives per heat of steel:

	Calories.
From hot producer gas at 600°	= 831,980
From preheated air at 1,000°	= 845,130
From combustion	= 4,389,700
Total	= 6,066,810

Of this total there is rejected in the hot products at 1,400°, 3,816,400 Calories, leaving in the laboratory of the furnace 2,250,410 Calories.

According to the view of Damour and Queneau (Industrial Furnaces, page 56) the ratio of the heat thus left in the body of the furnace to the calorific power of the fuel used, measures the thermal efficiency of the furnace. According to that view this new style Siemen's furnace has an efficiency of

$$\frac{2,250,410}{5,647,400} = 0.400 = 40.0 \text{ per cent.}$$

This figure, however, is an illusive one. It enables us to compare two furnaces and see which regenerates the waste heat best, or which furnace laboratory is designed best so as to catch and retain most of the heat furnished to it; but the real question is the comparison of different laboratories as to their net melting efficiency. This laboratory abstracts from the heat supply furnished to it 2,250,410 Calories, and furnishes to the steel 900,000 Calories. Its proportion of usefully applied heat to heat appropriated is, therefore:

$$\frac{900,000}{2,250,410} = 0.400 = 40.0 \text{ per cent.} \qquad (3)$$

The laboratory of the furnace therefore loses by radiation 60 per cent of the heat which it abstracts from the gases, or 2,250,410 — 900,000 = 1,350,410 Calories. This is 24 per cent of the calorific power of the coal used.

(4) The regenerators receive 3,816,400 Calories from the laboratory of the furnace, and return to it the heat in the preheated air at 1,000°, viz.: 845,130 Calories. If we call the ratio of these two the efficiency of the regenerators we have

$$\frac{845,130}{3,816,400} = 0.222 = 22.2 \text{ per cent.}$$

If, however, we charge them only with the heat actually left in them by the hot products, entering at 1,400° and leaving at 350°, we have an efficiency of

$$\frac{845,130}{3,816,400 - 768,950} = 0.277 = 27.7 \text{ per cent.} \qquad (4)$$

As long as the chimney gases are near in temperature to 300°, we can use the latter form of calculation as representing the real efficiency of the regenerator. The regenerators, therefore, lose by radiation and conduction to the air 2,202,320 Calories, which is 72.3 per cent of the heat left in them by the hot gases and 39 per cent of the calorific power of the coal used.

(5) The flame temperature is that to which the 6,066,810 Calories available in the laboratory of the furnace will raise the products of combustion. The average mean specific heat of the latter per 1° is:

CO^2	986 m^3 ×	(0.37 + 0.00022t)	= 365 + 0.2169t
$O^2 + N^2$	4,203 m^3 ×	(0.303 + 0.000027t)	= 1,274 + 0.1135t
H^2O	1,135 m^3 ×	(0.34 + 0.00015t)	= 386 + 0.1703t
		Sum	= 2,025 + 0.5008t

Therefore $$t = \frac{6,066,810}{2,025 + 0.5008t}$$

Whence $$t = 2,003° \qquad (5)$$

Problem 72.

In an basic lined open-hearth furnace using the Monell process, 50 short tons of melted pig iron, at 1,300°, is run upon 30,000 pounds of Lake Superior iron ore (90 per cent Fe^2O^3, 10 per cent SiO^2) lying on the hearth and previously heated to 1,300°. There is 2,000 pounds of burnt lime lying on the ore to help to form slag. The ore reacted quickly, almost violently, on the melted pig iron, so that at the close of the reaction, in 20 minutes, the slag could be run off. The pig iron contained.

	On Running in.	*After the Reaction.*
Carbon	3.50	3.00
Silicon	2.00	0.09
Phosphorus	0.75	0.00
Manganese	0.50	0.00
Iron	93.25	97.00

The oxidation of C, Si, P and Mn may be assumed as being produced first by reducing all Fe^2O^3 present to FeO, and completed by the reduction of some FeO to Fe. The carbon is assumed oxidized in the reaction to CO, although this CO may be subsequently partly burned to CO^2 by excess air in the furnace.

Required:

(1) The amount of iron reduced into the bath.

(2) The weight and percentage composition of the slag formed.

(3) The items of heat evolved and absorbed in the reaction.

Solution:

(1) We cannot make a simple, direct solution, because the weight of the metal after the reaction, analysis of which is given, is not known. The inexperienced calculator might be tempted to say that there was 3.50 — 3.00 = 0.50 per cent of carbon oxidized, 2.00 per cent of silicon, 0.75 per cent of phosphorus and 0 50 per cent of manganese, calculate the weights of these, reckon up the oxygen they would absorb in being oxidized, and thus get at the amount of Fe^2O^3 reduced to FeO and FeO reduced to Fe. This would be correct as far as silicon, manganese and phosphorus are concerned, because they are entirely oxidized, but incorrect for the weight of carbon, because the 3.00 per cent not oxidized is per cent of the final bath, which is of different weight from the original one, and, therefore, we are in error in subtracting 3.00 from 3.50. Being confronted with this dilemma we can see, however, that if we only knew the weight of the bath after the reaction we could calculate correctly how much carbon is in it, thence get the correct weight of carbon oxidized, then the correct amount of oxygen absorbed in the reaction, and from this the weight of iron reduced into the bath. We could also get the latter quantity more simply by finding the iron present in the bath after the reaction (97 per cent), and subtracting from it the iron in the original bath. We thus have two ways of getting the same requirement if we only knew the weight of the bath. In such a case the mathematical key to the situation is, of course, to let X represent the weight of the bath, and work out the amount of iron reduced into the bath by the two methods,

getting the result expressed in each case in terms of X. Since the two expressions represent the same quantity, we put them equal to each other and solve for X. Having obtained X, we substitute it in other expression and get the result asked for.

Let X be the weight of the bath after the reaction, in pounds; the bath before the reaction weighs 100,000 pounds.

Oxidized out:

Carbon (100,000 × 0.035) — X 0.03	= 3,500 — 0.03 X	pounds.
Silicon 100,000 × 0.02	= 2,000	"
Phorphorus 100,000 × 0.0075	= 750	"
Manganese 100,000 × 0.005	= 500	

Oxygen required:

For carbon (3,500 — 0.03 X) 16/12	= 4,677 — 0.04 X	pounds.
For silicon 2,000 × 32/28	= 2,286	"
For phosphorus 750 × 80/62	= 968	"
For manganese 500 × 16/55	= 145	"
	Sum = 8,076 — 0.04 X	pounds.

Oxygen supplied:

If all Fe^3O^3 of ore (27,000 pounds) were reduced to FeO = 27,000 × 16/160 = 2,700 pounds.

If all Fe^3O^3 of ore were reduced to Fe 27,000 × 48/160 = 8,100 pounds.

We thus see that we will certainly require more oxygen than the Fe^3O^3 can give up in becoming all FeO, but not as much as would be given up if it all became Fe. If all the Fe^3O^3 were considered first reduced by the reaction to FeO, giving up 2,700 pounds of oxygen, the reaction will still require the furnishing of

$$(8.076 - 0.04\ X) - 2,700 = 5,376 - 0.04\ X \text{ pounds}$$

of oxygen, which would have to be furnished by FeO becoming Fe. In that reduction, however, 72 parts of FeO gives up 16 of oxygen in becoming 56 Fe. The reduced Fe will be, therefore, 56/16 of the weight of oxygen thus furnished, and therefore,

$$\text{Reduced Fe} = 56/16\ (5,376 - 0.04\ X) \text{ pounds.}$$
$$= 18,816 - 0.14\ X \text{ pounds.}$$

The same quantity is obtained more directly as follows:

Fe in original bath 100,000 × 0.9325	= 93,250	pounds.
Fe in bath after reaction	= 0.97 X	pounds.
Therefore, Fe reduced	= 0.97 X — 93.250	lbs.

These two expressions represent the same thing, and, therefore,

$$18,816 - 0.14\ X = 0.97\ X - 93,250$$

Whence $X = 100,960$

And the reduced iron = 0.97 X — 93,250 = **4,681** pounds. (1)

(2) The ore used contains altogether

Fe = 27,000 × 112/160	= 18,900	lbs.	
Fe reduced to metallic state	= 4,681	"	
Fe remaining in slag as FeO	= 14,219	"	
Weight of FeO = 14,219 × 72/56	= 18,282	"	= 61.1 per cent.
Weight of P^2O^5 = 750 + 968	= 1,718	"	= 5.7 "
Weight of MnO = 500 + 145	= 645	"	= 2.1 "
Weight of CaO	= 2,000	"	= 6.7 "
Weight of SiO^2 = 3,000 + 4,286	= 7,286	"	= 24.4 "
Total weight of slag	= 29,931	"	(2)

(3) The physical data available do not permit of calculating the actual heat of the reaction at 1,300°, since many of the specific heats needed are lacking. We will therefore foot up the items of heat evolution and absorption uncorrected for temperature, which is the only thing to be done under the circumstances.

Heat Evolution.

		Calories.
Si to SiO	2,000 × 7,000	= 14,000,000
P to P^2O^5	750 × 5,892	= 4,419,000
Mn to MnO	500 × 1,653	= 826,500
C to CO	471 × 2,430	= 1,144,500
SiO^2 to FeO. SiO^2	7,286 × 144	= 1,049,200
CaO to 3CaO. P^2O^5	2,000 × 949	= 1,898,000
	Total =	23,337,200

Heat Absorption.

		Calories.
Fe^3O^3 to FeO	18,900 × 573	= 10,829,700
FeO to Fe	4,681 × 1,173	= 5,490,800
Fe^3C to Fe^3 + C	471 × 705 (?)	= 332,000 (?)
FeSi to Fe + Si	2,000 × 931 (??)	= 1,682,000 (??)
Fe^3P to Fe^3 + P	750 × 1,400 (??)	= 1,050,000 (??)
	Sum =	19,564,500

These calculations, therefore, show a minimum surplus of heat in the reaction of 23,337,200 — 19,564,500 = 3,772,700 Cal., an amount which would raise the temperature of the slag and resulting metal approximately 100° C. above the 1,300° at which the reacting materials came together. The quantity above marked (?) is a little doubtful in amount, but those marked (??) are very doubtful, perhaps may not exist at all. If these are omitted from the heat absorption the surplus heat is increased some 50 per cent of its value, and the rise in temperature might be in the neighborhood of 150°. Further, some of the CO formed by the oxidation of carbon may be burned to CO^2 close to the surface of the bath, by free oxygen in the furnace, still further increasing the rise in temperature.

The conclusion from these calculations and discussions is that the pig iron and ore reaction in the Monell process is a heat evolving reaction, which, independently of the heating effect of the fuel used by the furnace, could increase the temperature of the contents of the furnace at least 100°, and possibly 150°. It would be highly interesting to have a typical heat such as this followed closely with a good reliable pyrometer, so as to check the indications of the thermochemical study of the process.

[The next instalment of these calculations will be upon the electrical methods of melting iron and steel, electro-thermal processes of reducing iron ore and the electrolytic refining of iron.]

Magnetic Separation of Iron Ores by the Gröndal Process.

By P. McN. Bennie.

Development of the Gröndal Process.

About twenty years ago the question of iron ore supplies became serious, and at that time magnetic separation received considerable attention from Conkling, Ball and Norton, and others, with the idea of utilizing large bodies of magnetic iron ore which could only be made available by means of magnetic concentration.

Then came the rapid development of the immense bodies of Lake ore, of a quality which could advantageously be used in the blast furnace, and, except in certain restricted districts, interest in magnetic ores was lost. About the same time there was some hope that the magnetic iron ore deposits in Ontario would receive serious development, and indeed a number of fairly large shipments were made to the United States. But the development of the Lake ore bodies practically closed that market, and work on the Ontario mines declined to insignificance. In fact the discovery of those same Lake Superior ore bodies has had an influence upon the metallurgical life of both Canada and the United States, but little appreciated nowadays.

In European countries, however, especially in Sweden, conditions were different. The supply of suitable ores ready

*A paper read before the Canadian Mining Institute, slightly abstracted.

for blast-furnace use in their natural conditions was limited and low grade; in many cases highly phosphatic and sulphurous ores had to be used; consequently great efforts have been made to perfect concentrating methods. The attention of many prominent engineers and metallurgists has been turned, with more or less success, towards the solution of the difficult problems involved in turning to account these ore supplies.

Mr. Gustaf Gröndal's efforts in this direction, extending over a number of years, have been quite successful, the ap-

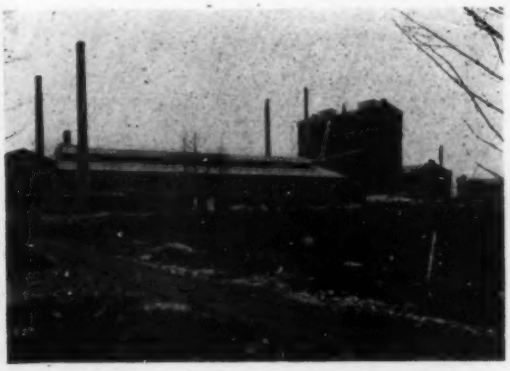


FIG. 1.—GENERAL VIEW OF HERRÄNG IRON WORKS.

pliances and machinery invented by him for crushing, concentrating and briquetting low grade and impure iron ores being now so perfected as to constitute a complete commercial process. By these methods a great many hitherto practically valueless iron ores can be turned to excellent account by reason of the richness and purity of the concentrates and briquets produced, and the comparatively low cost of production.

Primarily the Gröndal processes permit of the economic use of the many, in some cases immense, deposits of iron ore which are found in various places throughout the world, and which are unsuitable in their native state, owing to some objectionable characteristic, for economic blast-furnace practice, and the production of good quality pig iron. Usually, the objection to their use will be found to be one of the following: First, that the ore is too low in iron content; second, that it contains impurities which cannot be eliminated in the blast furnace; third, that it occurs in a mechanical condition unsuitable for blast furnace use, except to a limited extent: such as iron sands and small ores generally.

Among the ores which may be commercially utilized by the Gröndal processes may be instanced:

Magnetites of all kinds, including magnetic iron sands, containing as low as 25 per cent iron.

Magnetites containing a high percentage of phosphorus and copper not chemically combined with the iron.

Purple ore and pyrites residues.

Nearly all kinds of iron ores containing a high percentage of sulphur.

Flue dust.

The Gröndal Concentrating Process.

The scheme of treatment is briefly as follows:

1. The ore is crushed dry to about ½-inch cube or thereabouts.

2. Wet treatment in a Gröndal ball mill, which reduces the ore from the crushed size to 10 to 100-mesh as may be found necessary.

3. The ground pulp is passed through a Gröndal magnetic separator where the non-magnetic particles are removed.

4. The concentrates are pressed into briquets and heated in the Gröndal briquetting kiln, from which they issue as hard, porous, easily reducible, ferric oxide briquets, with a minimum percentage of sulphur.

To take each operation in detail:

Any good type of commercial crusher may be used in the preliminary crushing of the ore. Usually either a Gates or Blake crusher is employed.

The broken ore is conveyed and distributed to the feed hoppers of the Gröndal ball mills. The feeding arrangement consists of an ordinary roller-feeder driver at a uniform speed and so constructed as to admit of accurate adjustment of feed. The ball mill consists of a hollow cylinder, 4 feet in diameter and varying in width from 4 to 8 feet. It is reinforced with steel ribs and lined with manganese steel, or other steel alloy of good wearing quality. The mill is charged with about 2 tons of chilled cast-iron balls, of about 6 inches diameter.

The ends are of cast iron, with suitable openings provided with a screen at the discharge end. The degree of fineness of the ore is regulated by varying the amount of water introduced into the mill.

It is found that working with ores of average hardness the consumption of iron, represented mainly by the wear on the balls, amounts to about 2 pounds per ton of ore treated.

The mills require from 20 to 25 hp. each; make from twenty-five to thirty revolutions per minute, and grind from 50 to 100 tons of ore per 24 hours, from ½ inch down to from 10 to 100-mesh in fineness.

In order to free the pulp from the bulk of the non-magnetic climes, it is often advisable to pass the pulp coming from the ball mill through a slime-box before charging it into the separator proper. The slime-box consists of a V-shaped box, the plup entering at the top and a stream of water at the bottom. The slime boxes are usually built in pairs, and between each

FIG. 2.—BALL MILLS AT HERRÄNG.

pair is placed an electromagnet with a hatchet-shaped pole piece. This electromagnet is of such strength as just not to lift any magnetic particles out of the water. The velocities of the pulp and clear water are so arranged that the heavier particles settle at the bottom, from whence they are drawn off to the separator. The function of the magnet is to arrest magnetic particles in the fine slimes which collect on the surface of the water immediately under the pole piece in masses,

and from time to time drop to the bottom whence they are carried off with the main pulp to the separators.

The diagram of Fig. 3 shows the principle of the separator proper. S representing the slimes, M the magnet, C the concentrates, T the tailings. The pulp feed from slimes passes to this part of the apparatus. This consists of a series of magnets with flat pole pieces arranged with their N and S poles alternately. The magnets are enclosed in a plain brass drum. This drum is made to rotate at a speed of from 80 to 100

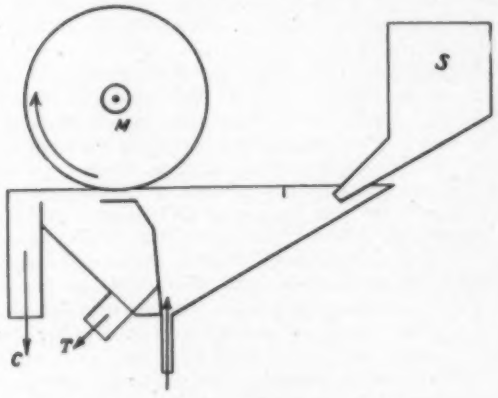


FIG. 3.—PRINCIPLE OF GRÖNDAL SEPARATOR.

revolutions per minute, and about 1 inch above the surface of the pulp, which traverses a pyramidal box divided into two compartments by a partition reaching nearly to the top of the box. The pulp enters at the top of the box, and a stream of water entering at the bottom and rising up on the same side of the partition carried all the pulp well over the edge of the latter and thus immediately under the drum.

The magnets within the drum powerfully attract and bring the magnetic particles out of the stream of pulp and against the revolving brass drum. The remainder of the pulp drops down on the other side of the partition, and is carried off by the water through the waste outlet. The magnetic particles are lifted by the drum to the very edge of the magnetic field, where they are thrown off centrifugally. The capacity of the double-drum separator (which is the form commonly used) is from 70 to 100 tons of ore per 24 hours; it requires about 6 amps. at 120 volts, or, say, 1 hp., while less than ¼ hp. will drive the drums.

A plant consisting of four ball mills, four double-drum separators, and having a capacity of fully 200 tons of concentrates per 24 hours, would require approximately 180 gallons of water per minute, and a power plant generating 150 electric horse-power would be ample for all the motive power required.

The concentration process reduces the percentage of phosphorus more or less, depending upon the mineral form in which it occurs in the ore. In most magnetites it is found in the form of apatite, in which case the phosphorus can be almost entirely eliminated. For example, at Gellivare, Sweden, the phosphorus is reduced from 1.29 in the crude ore to 0.005 in the concentrates, and 0.006 in the briquets. The percentage of iron is raised to a high figure, and at the same time the amount of slag-forming minerals is lowered. As a result the ore is more easily reduced in the furnace, and requires less fuel per ton of iron.

The quality of the pig iron will be quite materially improved when employing purer ore and less fuel than ordinarily.

Sometime the ore mined is a mixture of magnetite and hematite. The dry magnetic concentrating methods hitherto employed give rather unsatisfactory results as regards the iron content of the tailing. With the Gröndal concentrating process, where the ore is already slimed up in water, a retreatment of the tailings on jigs or slime tables can be carried out with advantage, thus increasing the output of the plant without crushing or grinding any additional tonnage of ore.

The Gröndal concentrating process can be regarded as practically the only existing method for the economic treatment of low-grade ores in which the magnetite is so finely and intimately mixed with the gangue that the ore has to be crushed down to less than twenty mesh before it can be separated. Take the case of the banded jasper ore in the Temagami district, which would require very fine grinding to effect an economic recovery of iron: the Gröndal process could do this, where no other could. But it is also obvious that such fine ores must be briquetted.

The Gröndal Briquetting Process.

Concentrates produced by this process, iron sands, pyrites residues, small ores generally, as well as ores containing a high percentage of sulphur, which are unsuited for use in the blast furnace, are next briquetted.

The material is conveyed to the briquetting presses. These are drop presses, the height of the drop being from 6½ to 7½ inches, and the briquets receive three blows, the falling weight being about 1,800 pounds. The briquets may be of various sizes, but are usually made 6 inches x 6 inches x 3 inches, weighing from 8 to 10 pounds each. Each press requires 3 electric horse-power, and will make from 500 to 750 briquets per hour. The pulverized ore is pressed into briquets without any binding material whatever, the moisture in the ore being so adjusted as to obtain briquets sufficiently firm to be removed from this press to the cars used in the furnace.

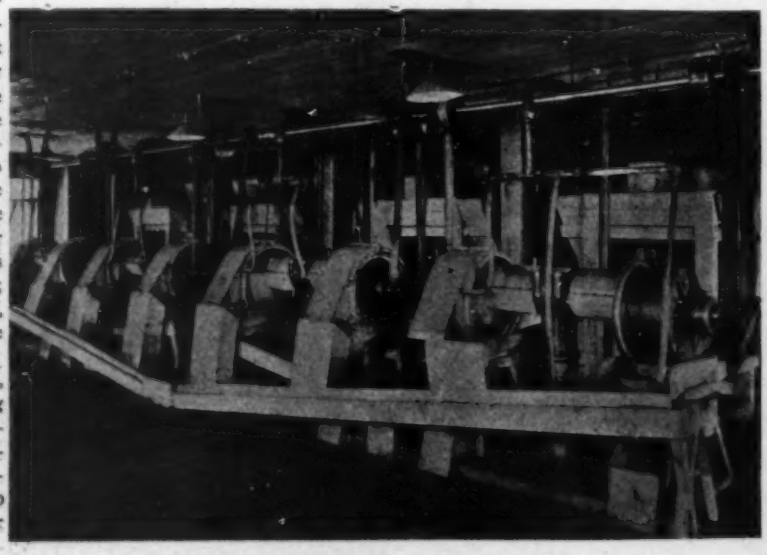


FIG. 4.—MAGNETIC SEPARATORS AT HERRÄNG.

The briquets are taken from the press and placed on the furnace cars. These are made of iron and covered with firebrick. Along each side of the car is a deep flange, which dips into a channel filled with sand placed along the sides of the furnace, thus forming a gas-tight seal. The ends of adjacent cars are fitted with a groove and projecting rib respectively. By this means the surface of the cars forms a gas-tight partition and thus prevents the power portion of the cars, frame, wheels, etc., from becoming heated. The cars measure 3 feet 6 inches,

and hold from 15 to 16 hundredweight of briquets, arranged on edge in two piers.

The Furnace.

The furnace is fired by gas derived preferably from gas producers, although blast furnace gas can be used. The combustion chamber is situated about one-third of the length from the in-take end. The air needed for combustion is introduced below and traverses the row of cars, thus helping to keep the wheels and framework cool, and at the end of the furnace is diverted to the top of the row of cars, traversing

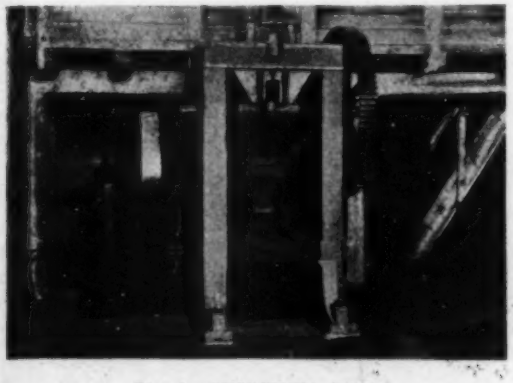


FIG. 5.—BRIQUETTING PRESS.

the burnt briquets, cooling them and becoming itself heated before reaching the combustion chamber. The products of combustion pass over the entering cars, assist in drying and heating the briquets, and finally escape through a stack at a temperature of only 150° C.

The furnace being thus constructed on the regenerative principle, has a very good thermal efficiency, radiation is small, and the chief loss of heat is due to the evaporation of the water contained in the briquets (5 to 7 per cent). A car of finished briquets is drawn about every half hour, depending somewhat on the degree of desulphurization required. Recently furnaces have been arranged so as to give a continuous movement to the cars, increasing the capacity of the furnace considerably.

The temperature in the combustion chamber of the briquetting furnace reaches 1,300° to 1,400° C. At this heat the particles agglutinate sufficiently to form a firm, hard briquet, able to stand rough handling and long transport. The briquets though hard are very porous, and are, consequently, far more easily reduced in the blast furnace than ordinary lump ore. The briquet made from Herräng ore, exhibited here, has a porosity of 23.9 per cent of its volume.

When briquetting iron ore concentrates and using producer gas for fuel, the consumption of coal has been found to average 7 per cent of the weight of briquets burnt.

Gröndal Briquets.

The Gröndal briquet is unique in that it does not contain an added binder, and therefore there is no foreign material such as lime to be heated and slagged off. In the case of magnetic oxide concentrates the particles are completely peroxidized, owing to the very free supply of air given to the furnace during the process of burning the briquets. Such peroxidized ore is very easily reduced in the furnace, thus introducing a certain fuel economy.

Treatment of Pyrites Residues.

Briquets made by this process from pyrites residues and purple ore have proved to be entirely satisfactory in the open-hearth steel furnace. At Cwmavon ores containing 2 to 4 per cent sulphur, and sometimes more, are being successfully treated by this process.

Capacity of Furnace.

The output of one furnace varies from 30 to 80 tons in 24 hours, according to the class of ore used and the degree of desulphurization required; for in addition to its mechanical action this furnace acts as an exceptionally efficient calciner for removing practically all of the sulphur.

Results of Treatment.

The following analyses of crude ore concentrates and briquets give an idea of the results obtained by the Gröndal processes. In addition to the figures, given below for the results obtained at Herräng, it may be said that the tailings from the Herräng ore, after passing through the magnetic separator, analyzed 9.6 per cent iron. The porosity in per cent of volume of briquets was 23.9:

	Iron.	*Sulphur.*	*Phosphorus.*
Herräng, Sweden—			
Crude ore	40.00	1.20	0.003
Concentrates	65.20	0.17	0.0025
Briquets	63.01	0.003	0.0025
Gellivare, Sweden—			
Crude ore	58.96	0.036	1.29
Concentrates	72,38	0.003	0.005
Briquets	69.49	0.002	0.006
Salengen, Norway—			
Crude ore	36.43	0.021	0.318
Concentrates	71.76	0.015	0.008
Briquets	....	....	....
Pitkaranta, Finland—			
Crude ore	28.40	2.60	0.260
Concentrates	69.59	0.132	0.007
Briquets	67.98	0.011	0.008
Cornwall, Penn.—			
Crude ore	30.65	1.603	0.012
Concentrates	69.95	0.036	0.003
Briquets	69.90	0.010	0.005

The results from Cornwall ore were obtained with a 10-ton

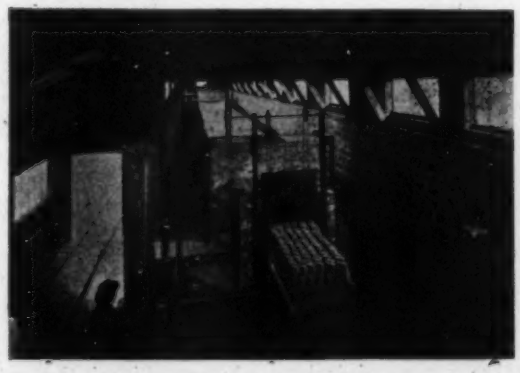


FIG. 6.—BRIQUETTING FURNACE AT GULDSMEDSHYTTAN.

sample sent to Herräng about a year ago, as there is not yet a plant available on this side of the Atlantic.

Separation of Titanium.

Last June we constructed a small separator on the Gröndal principle at our Canadian laboratory. Some tests were made on the separation of titanium from ore, working with what are known as Moise Beach Sands, from the St. Lawrence River. We got the following results:

Original	16.57 per cent TiO_2
Concentrates	4.21 "
Middlings	19.48 "

It should be remarked that the concentrates were obtained by a single passage through the separator. Doubtless even better

results could be obtained from a finer grinding, and repassing through the separator.

Cost of Plant.

It is impossible to give anything but bare approximations as to cost of plant, which varies considerably with the local situation and conditions. The figures quoted are based upon cost of materials and cost of erection in the United States:

For 200 tons of concentrates per day:

Concentrating plant for 400/500 tons of crude ore per day	$45,000
Briquetting plant for 200 tons of concentrates in 24 hours	65,000
	$110,000

For 100 tons of concentrate per day these figures would be about $27,000 and $39,000, respectively, a total of $66,000.

Cost of Concentrating.

The cost of concentrating will naturally vary with the character of the ore treated, and the amount of iron contained in same. Assuming the treatment of an average magnetic iron ore, containing 40 per cent iron, and the production of a concentrate carrying about 63 per cent iron, the cost of concentrating varies from 30 to 45 cents per ton of concentrates. These figures refer to the mechanical treatment only and exclude the cost of ore.

Cost of Briquetting.

The cost of briquetting is also an item varying with local conditions. A briquetting plant of 200 tons daily capacity would consist of four to seven furnaces, depending upon the ore, and to a large extent on the amount of sulphur to be removed. The cost per ton for briquetting can be taken, under normal conditions, as varying between 45 and 85 cents per ton.

At a small plant consisting of one furnace, producing 25 to 30 tons per day, the actual cost, including labor, coal, rent and taxes, but exclusive of depreciation and royalty, amounts to 85 cents. This plant is working on pyrites residues, and owing to its small size and the nature of the work the cost is very high.

On the other hand, at a typical plant in Sweden, turning out 120 tons of briquets daily, the cost of briquetting amounts to 77 cents per ton. At one of the Swedish plants of recent construction the cost has been reduced to 67 cents per ton.

Recently the briquetting furnace has been simplified and improved so as to materially decrease the cost of installation, and at the same time to lower the cost of briquetting.

Treatment of Canadian Ores.

The present situation in Canada, and particularly in the province of Ontario, is similar to that in the Scandinavian countries in many ways. Not over blessed with ready fuel, and with iron ores that in many instances require concentration, yet it is well known that certain qualities of iron (Swedish pig, for example) are produced, which bring a premium on the market. There is no reason why Canada should not largely augment its production of high-grade pig iron, and have as great a demand for it as for the Swedish article.

There are many bodies of magnetic iron ore in Ontario, running from 35 to 50 per cent iron, that would be so improved by proper treatment as to produce premium ores, for which there is now, and probably will continue to be, a brisk demand.

For ores that demand fine grinding to recover an economical percentage of iron content, the Gröndal briquet affords a means of converting unsalable ores into an article that will actually bring a premium, thus helping to pay for the cost of treatment. Only recently a blast furnace manager told me he would be glad to get 200 tons a day of such briquets. Gröndal briquets are now being shipped in considerable quantities from Sweden to Great Britain, where they command an instant market at prices above the average.

Electrothermic Processes.

Another feature certain to be of considerable economic value to Canada is the application of electrothermic processes for producing high-grade pig iron and steel. Broadly speaking, the use of sufficient electrical energy to produce 1 ton of pig iron in the electric furnace means the displacement of about 1,600 pounds of fuel. Coal once used is gone forever, while water powers maintain themselves by a natural process, a fact of great economic importance to Canada. The fortunate existence, practically side by side, of iron mines and extensive water powers, produce conditions very favorable to the electrothermic process. I think the Government would do well to foster the development of that branch of metallurgy as much as possible, to enable the country to realize on its natural and unique resources.

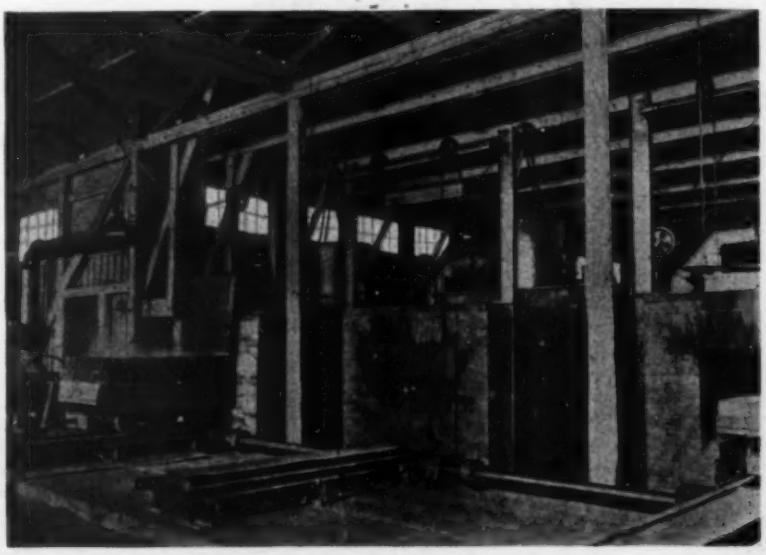


FIG. 7.—RANGE OF BRIQUETTING FURNACES AT HERRÄNG.

Many of the ores could doubtless be used direct, others would require concentration, and still others would have to be briquetted, but the economy of producing finished material wherever possible "on the spot" is obvious. Science and art have now placed in the hands of those who will, means of utilizing these resources, and thus adding to the nation's wealth.

FitzGerald & Bennie Laboratories, Niagara Falls.

Aluminium.—Like the Aluminum Co. of America (which is the new name of the Pittsburg Reduction Co.) the British Aluminium Co. is making large extensions. They have acquired a partly-developed water power at Stangfjord, in Norway, and have taken steps to develop speedily some 3,000 hp at Loch Leven. They expect to produce aluminium in Norway within about two months and at Loch Leven in Autumn. In addition, they have purchased the concessions for a water-power at Orsieres, in Switzerland. It is expected that when all these schemes are fully developed the company will have upwards of 70,000 hp. per year.

The Open-Hearth Steel Process.

Engineer Theodor Naske contributes to *Stahl und Eisen*, of Jan. 30, Feb. 6, 13 and 20, a long discussion of the reactions in this process. He starts out with the rather despairing remark, that the processes going on in the open-hearth furnace are practically uncontrollable; this would hardly be a statement which is true, but it happily turns out afterwards that the writer means that it is difficult to account for the influence of mass and temperature, and therefore hard to predict in which direction the reactions will proceed—nothing more serious than that. In the first place, the author sets out to determine how and in what way the ingredients of the charge are oxidized into the slag or reduced back therefrom into the metal.

(1) The behavior of manganese as a conveyor of oxygen. The circumstances under which this element acts as an oxidizing agent upon the bath or as a reducing agent upon the slag, are to be found in the temperature, the chemical composition of the slag, and in the concentration of the manganese in the slag and in the metal. When manganese is oxidized from the bath it goes into the slag first as MnO, forming ferrous-manganese silicate, the slag evidently striving towards the formation of uni-silicate FeMn, SiO^4. By a comparison of thirty-two sets of analyses of metal and slag at the beginning and ending of the refining period, it was determined that whenever the manganese in the slag was less than the iron the bath lost manganese, whenever the manganese equalled the iron the bath neither lost nor gained, whenever the manganese exceeded the iron there was manganese reduced from the slag and the bath gained manganese. This was independent of the total percentage of iron and manganese in the slag, but concerned only their relative proportions towards each other. [It follows that, after the slag contains as much manganese as iron, the bath ceases losing manganese, even in an oxidizing atmosphere, and that further oxidation of manganese can only go on either as the slag receives iron oxide from oxidation of the bath (Fe and Mn would then oxidize from the bath in equal quantities), or as iron oxide is bodily added to the slag from an outside source.] Naske also draws the conclusion that a slag containing less manganese than iron not only oxidizes manganese from the bath, but usually co-exists with a bath containing dissolved FeO; that the presence of manganese equal to iron in the slag stops the solution of FeO in the bath, or even tends to remove FeO and thus purify the bath. In this respect, manganese as used systematically in the Terre Noire process, for instance, is a valuable and almost indispensable agent in the slag for preventing iron oxide being taken up in large quantities by the bath.

(2) The behavior of silicon and carbon. The silica in the slag depends on the silicon in the pig iron, that in ore additions and that from furnace lining; and the quantities of these going into the slag are the most important factors in determining the amount and quality of the slag. Carbon is more easily oxidized by artificial oxidizing agents than by free oxygen, so that there is no doubt that the carbon removed from the bath is in reality oxidized by metallic oxides. Carbon is not oxidized while silicon is present in large amount, because silicon can reduce any carbon monoxide which tends to form, producing SiO^2 and sending the carbon back into the bath. $Si + 2CO = SiO^2 + 2C$. We may assume that both carbon and silicon oxidize, but the above reaction prevents any carbon monoxide escaping. The carbon is only finally oxidized when the silicon is reduced to such proportion in the bath that, according to the law of mass action, its activity in reducing carbon monoxide is less than the speed of formation of the latter. On the other hand, it has been established that from acid slags, containing up to 55 per cent SiO^2, silicon may be reduced back into the bath at a high temperature by the carbon dissolved in the iron.

(3) The behavior of phosphorus. Two tests were carefully made to determine the action of cold ore or hot ore on pig iron to remove phosphorus. In one test 3,280 kg. of ore was dumped upon the empty hearth, 984 kg. of limestone upon it, and immediately, while these were yet cold, 20,303 kg. of pig iron run in upon them. The tests show very plainly the nature of the reaction:

	C.	*Si.*	*Mn.*	*P.*	*S.*
Metal used..	4.61	0.84	2.20	0.15	0.02
Bath in 20'..	4.56	0.19	0.45	0.05	0.01
Bath in 40'..	3.86	0.09	0.45	0.03	0.02

	Fe.	*SiO^2.*	*Mn.*	*P^2O^5.*	*FeO.*	*Fe^2O^3.*
Slag at 20'..	41.51	17.68	15.22	2.36	47.88	6.10
Slag at 40'..	31.67	19.05	15.71	2.93	36.29	4.91

For comparison with this, 3,280 kg. of ore was charged with 820 kg. of limestone into an empty furnace, and strongly heated for 30 minutes, until it was at bright red heat but not melted. A sample taken from the furnace showed none of the ferric oxide changed to Fe^3O^4 by this heating. Upon this 19,730 kg. of fluid pig iron was run, reproducing the Monell process reaction. The reaction was very violent, and in 25 minutes showed the following changes to have taken place:

	C.	*Si.*	*Mn.*	*P.*
Metal used..	3.81	0.79	1.95	0.17
Bath in 25'..	3.62	0.09	0.31	0.02

	Fe.	*SiO^2*	*Mn.*	*FeO.*	*Fe^2O^3.*
Slag at 25'..	19.88	16.90	12.62	23.90	1.86

Still another similar test was made with 3,280 kg. of ore and 820 kg. of limestone, but these were heated until at the end of an hour they were completely melted to a homogeneous mass. A test of this showed it to be strongly magnetic and the iron mostly present as Fe^3O^4. The fluid pig iron, weighing 17,843 kg., was then run in slowly, and reacted violently, with great evolution of gas, showing the carbon to be rather strongly attacked. Test and analyses showed the following reaction:*

	C.	*Si.*	*Mn.*	*P.*
Metal used..	3.90	1.05	1.56	0.14
Bath in 15'..	3.35	0.05	0.28	0.01

The three heats just noted took respectively 4 hours 50 minutes, 4 hours, and 3 hours 10 minutes, showing the great advantage of operating according to the Monell process.

The removal of phosphorus is complicated by the fact that either carbon or carbonous oxide gas can reduce phosphorus from its compounds at a comparatively low temperature. In general, however, raising of the temperature favors removal of phosphorus from the metal; the dephosphorization is only possible in presence of iron oxide or lime in the slag; the solution of phosphoric acid in the slag adds greatly to the intensity of the reaction; an iron-lime silicate favors most the elimination of phosphorus; manganese in the bath exerts to influence upon the conditions of removal of phosphorus; the slag has a certain saturation limit for phosphoric acid, determined by its composition and the temperature, above this limit phosphorus may be reduced back into the bath.

The author is of the opinion that almost all the oxidation of elements other than carbon from the bath is effected indirectly by the carbon oxidizing first and then the gas acting upon the impurities, which latter take the oxygen and send the carbon back into the metal. In this view, Engineer Naske is certainly wrong, for his own facts and figures, closely studied, prove directly the contrary. It would be much nearer the truth, and would agree with all the facts adduced, to say that all the elements except iron are probably in largest part oxidized by the dissolved FeO, taken up by the bath, because of the overwhelming amount of iron present, and then reduced back to Fe as it oxidizes the carbon, silicon, manganese and phosphorus present. Naske makes a great deal of the application of the mass law, as largely governing the reactions in the open-hearth furnace, but has made the capital mistake of neglecting the overwhelming mass of the iron itself in the bath of metal being oxidized.

J. W. Richards.

* See calculations respecting this reaction in paper on Metallurgical Calculations by Prof. Joseph W. Richards, elsewhere in this issue.

Notes on Electrochemistry and Metallurgy in Great Britain.

(*From Our Special Correspondent.*)

PLATINUM SUBSTITUTES.

A company with the rather suggestive title of "Platinum Substitutes, Ltd.," has been registered with a capital of £50,000 in £1 shares, to acquire from C. O. Bastian and G. Calvert the benefit of certain inventions relating to sealing metallic conductors into or through glass, to turn the same to account, and to carry on the business of electricians, makers of cables, wires, lines, accumulators and lamps, suppliers of electricity, etc.

METALLURGICAL PAPERS READ IN FEBRUARY.

Three papers were presented to the Institution of Mining and Metallurgy for their meeting on Feb. 21. The first by Mr. W. R. Thomas on the "Electrically-Driven Centrifugal Pumping Plant at the Tywarnhaile Mine" appealed to mechanically-minded members. The second paper, by Mr. E. H. Garthwaite, discussed "Development Results versus Stope Results," and presents actual comparative diagrams and figures. In the case under examination the value of the ore was fairly uniform, and the discrepancies between the two methods of sampling were not great. The third paper, by Mr. C. F. Hughes, dealt with "Minerals and Metalliferous Lodes of Kulu and Lahol, Kangra District, Junjab, Himalayas." The chief features of Kulu are pyrrozhotite containing 15 to 20 ounces of silver per ton, and iron ore of high quality. It is pointed out that if communication were better, and a railway brought within reasonable distance, it might pay to set up an electric smelting plant to treat these ores. There is abundant water power and large forests. These ores have been smelted by the natives in small charcoal furnaces for several centuries. As regards Lahool, gold is extracted in very paying quantities by the natives from the sands of the Chandra River. A massive lode of pure stibnite 15 feet thick was reported some years ago. In view of the present scarcity of antimony it might be worth while to give more attention to this subject.

THE FARADAY SOCIETY.

It was unfortunate that the meeting on Feb. 19, to which Mr. J. B. C. Kershaw presented a paper upon the "Electrolytic Bleach and Alkali Industries" was somewhat sparsely attended. The discussion, however, occupied the whole evening; the chief features were the reminiscences of the early days of the industry contributed by Mr. Rhodin and by Mr. Richardson. The former, in particular, threw much light upon the evolution of the mercury cathode, and amused the audience by one or two rather pertinent asides aimed at the dissociation and other theories. Speaking on the subject of the voltage required for such processes, the observation was made that "if you were a plain, practical man, you found it to be four volts, but if you were a theorist you called it anything you liked." As to the fetters which restricted the development of such industries, they had to remember the evils of over capitalization. It was no good expecting to get rich in a few weeks as some speculative company promoters had anticipated, a successful industry required a slow and steady development. Then, too, inventors had been too optimistic—they had miscalculated capital requirements, and miscalculated wear and tear. Lastly, further extensions were hampered by the exorbitant value put upon waterfalls by their owners.

Mr. Richardson, speaking also in the reminiscent vein, gave a new version as an actual participator in the Richardson-Holland process.

A large number of questions were aimed at the author by the other speakers, who included Dr. Borns, Dr. Perkin, Dr. Seligman, Mr. Gaster and Mr. Digby. Some questioned points in the paper, but more asked for information concerning the merits of different makes of graphite and carbon, such as life and working current density, the life and material used for diaphragms in diaphragm processes, and so forth. The general feeling was that of regret at the extreme reticence of the manufacturers who had declined to give particulars of practical or any interest, leaving the author to base the greater part of his paper upon patent specifications. Upon the motion of Dr. Lowry, who presided, a vote of thanks was unanimously passed to Mr. Kershaw, who deferred his reply to the transactions of the Society.

MARKET PRICES.

The end of February finds the iron market depressed and wanting in tone. The collapse of the "bull" movement has emphasized the steady fall in Cleveland warrants, which are standing at 54.6. Steel rails are fairly steady at £6.15, and plates £7.10. per ton. Copper is still tending to go higher and occupies a firm position at £108. It does not seem probable that the supply will approach the demand for some months. Lead is again lower at £19.17 to £20 per ton. Tin has been lower, rallying to £193. Zinc is quoted at £30.15 per ton. Platinum remains at £7.10. per ounce. Shellac still at £11 per cwt. Para rubber is steady at 5.1. to 5.1½. per pound. Copper sulphate is quoted £32.10 per ton, no alteration on last month.

LONDON, March 5, 1907.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

ELECTRIC FURNACES.

Nickel Silicide and Ferronickel Silicide from Sulphide and Silicates.—Thomas L. Willson, 847,267 and 847,300, March 12, 1907. Application filed April 11, 1904.

Mr. Willson, whose pioneer work in the calcium carbide industry is so well known, applies in these patents his carbide arc furnace to the problem of treating sulphides or silicates of nickel, iron or copper for the production of silicides of these metals, either singly or alloyed. The process consists simply in smelting a mixture of the ore, lime, silica and carbon in an electric furnace. Practical applications are the treatment of the worthless form of iron silicate of sulphur, which is now a waste product in the reduction of nickel-sulphide ores; also the treatment of nickel and iron sulphide ores, which contain usually a small proportion of copper. If a mixture of the latter ore with limestone, silica and carbon were smelted in a blast furnace, there would result a calcium silicate of iron and a nickel-copper matte. But by the more active reduction obtainable by electric smelting calcium and silicon are freed, the calcium uniting with the sulphur of the ore and forming calcium sulphide, which floats off and the silicon uniting with the nickel and iron, forming a ferro-nickel silicide. Some of the silicon also combines with the sulphur to form silicon sulphide, which is volatile. To calculate the charge, one may proceed as follows: Enough lime is to be added, calculated to CaS, to combine with the total sulphur in the ore. Enough silica is to be added to form with the calcium equivalent thus determined $CaO(SiO_2)_2$. Then enough carbon is added to liberate from the calcium silicate the calcium and silicon, the former uniting with the sulphur and the latter with the iron and nickel. In practice, however, very much less lime, silica and carbon—perhaps only half the theoretical amounts found in the above manner—are sufficient. This is explained by the intense heat "dissociating sulphur, which is volatilized and

driven off and could doubtless be recovered as a by-product."

The following figures are given for the treatment of a ferro-nickel sulphide ore, containing 44 to 48 percent of iron, 4 to 7 nickel, 1 to 2 copper, 26 to 28 sulphur, 14 to 22 insoluble (diorite). To 300 pounds of this ore, suitably crushed, add 120 pounds of crushed limestone, 98 pounds of sand, and 107 pounds of granulated coke. This mixture is treated in an electric furnace, just like for the production of calcium carbide. Energizing the furnace with 35 to 37 volts and 4,000 amperes, the reduction of this quantity of material will require from 2 to 3 hours if started cold. The resulting ferro-nickel silicide has analyzed as follows: 75.6 per cent iron, 9.4 nickel, 12.46 silicon, 0.635 sulphur, 0.76 copper, 1.145 unascertained impurities. In running continuously, the calcium sulphide remains in the furnace when the silicide is tapped off, so that the quantity of lime or limestone used may be greatly diminished after the first few hours, the proportion of sand being correspondingly increased, thereby cheapening the process. For the more complete elimination of the sulphur the heat of the electric furnace may be increased, or the proportion of silica may be increased or fluorspar may be added, shortly before tapping off the product, introducing it preferably with a blast of air through tuyeres.

The process is also applicable to the treatment of silicates of iron, nickel and magnesium, such as those known as "New Caledonian" ores. In this case no silica need generally be added. The proportion of lime required may be diminished according to the proportions of magnesium contained in the ore. In the case of an ore containing an exceptionally large proportion of magnesium, due to its containing little or no free silica, the proportion of lime required may be reduced to the minimum. The proportion of carbon required is that which is required to combine with the oxygen compounds comprised in the ores.

The ferro-nickel silicide produced in this process is chiefly useful in the production of nickel steel. For this purpose iron is added to reduce the nickel to the desired proportion, which may be done either in the electric furnace or in a subsequent process, after which the silicon is eliminated by bessemerizing.

Production of Silicides and Silicon Alloys.—F. J. Tone, 842,273, Jan. 29, 1907. Application filed Dec. 16, 1905.

This patent reports further progress of the work of Mr. F. J. Tone, of the Carborundum Co. on the production of metallic silicon and silicon alloys. If, in the manufacture of silicides, for instance, manganese silicide, a charge of silica, manganese dioxide and carbon is heated in the electric arc furnace, an alloy or compound of silicon and manganese is obtained, but with a large loss of both silicon and manganese by evaporation. The inventor now states that this loss can be prevented by using an electric resistance furnace, with a molten bath forming the resistor and comprising the ore, metallic compound or metal with a reducing agent and solvent or flux, such as lime or fluorspar. For instance, when manganese and silicon are to be combined the charge is made up of 38 per cent of silica, 24 manganese dioxide, 21 carbon and 17 lime. This is heated in the crucible as shown in Fig. 1. The lower ends of the electrodes are immersed in the bath and the heat produced in it is sufficient to accomplish the reaction whereby the metal or alloy settles in the bottom of the receptacle as the layer 10, with the slag 9 overlying the metal. If metallic silicon is to be produced the charge consists of silicious ore with a reducing agent and flux. The metal, which is to be alloyed with the silicon, may, of course, be introduced into the charge as metal and not as oxide; for example, a silicide of iron and manganese may be obtained from a charge of ferro-manganese, silica, carbon and flux.

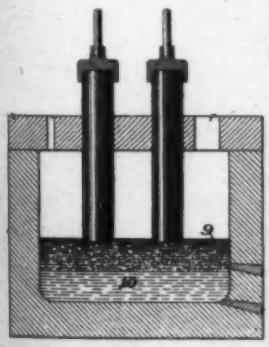


FIG. 1.—TONE ELECTRIC FURNACE.

Electric Furnace.—B. von Ischewsky, 847,003, March 12, 1907. Application filed Feb. 15, 1906.

The right-hand diagram is a vertical longitudinal section, and the left-hand diagram a vertical transverse section. The furnace chamber, containing the molten metal 2 is in form of a revolving drum lined with electric conductors of the second class, that is, such materials which do not conduct the current at ordinary temperature, but become conductors at high temperatures. The lining may be almost the same as in ordinary open-hearth furnaces or converters, being made of oxides, preferably magnesia, calcium oxide and other oxides, either basic or acid oxides. The carbon rods 3 are introduced radially from the outside through the walls of the drum, so as to just touch the oxide lining. These carbon rods can be pushed inwardly when they are burned away. They are connected by lateral pieces 5 to conductors 6 with the commutator 7, against which bear two brushes 8 and 9 through which the current is supplied. To start the furnace, hot slag or other hot material is introduced into the drum and the furnace then revolved somewhat, so that the inner layer is heated up to such an extent as to become electrically conductive. When the current is then switched on it passes from the carbon rods through the furnace lining. The curves of the current are denoted by the arrows in the left-hand diagram. Under all circumstances it is necessary that at least one electrode be outside of the metallic bath 2, so that the current is compelled to pass along the lining of the furnace. As special advantages of this design it is stated that the furnace can be operated directly by high-voltage currents. There are no carbon electrodes in contact with the charge; and when using consecutively electric furnaces with acid or basic linings any desired reactions for refining may be obtained.

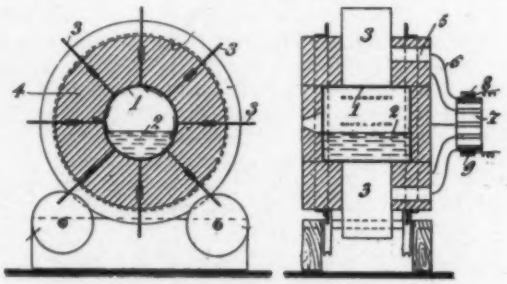


FIG. 2.—ISCHEWSKY ELECTRIC FURNACE.

Electric Furnace.—Le Roy Wright Stevens, 846,521, March 12, 1907. Application filed Sept. 16, 1905.

Details of mechanical construction of the mechanism by means of which a movable electrode is introduced through a furnace wall, so as to keep the furnace air-tight. The first claim relates to "the combination with the furnace body and the electrode of a stuffing-box, a movable sleeve mounted in the stuffing-box and surrounding the electrode, and providing a space between the inner wall of the sleeve and the electrode, granular refractory material in said space, means for retaining the granular material in position in the sleeve, and means for connecting the sleeve to the electrode." By means of water cooling the mechanism is protected against the heat of the furnace.

Production of Carbon Electrodes.—T. P. Sharts, 845,585, Feb. 26, 1907. Application filed March 6, 1905.

The object is to produce homogeneous carbon electrodes which are good conductors and at the same time withstand high temperatures. The nozzle of a burner extends through one

end wall of the furnace and hydrocarbon oils (petroleum) are burned under the influence of a blast of air under sufficient pressure so as to carry the atomized jet of mixed air and oil across the furnace chamber. The products of combustion are deposited on the opposite end wall of the furnace chamber, and to thicken the deposit this end wall is gradually removed to further distances from the burner. When the "charcoal-carbon" deposit is sufficiently thick, the wall is removed and the deposit is taken off and shaped into the desired form by cutting, grinding or otherwise. The product is said to be very hard and specially suitable for arc-furnace electrodes.

Electrolytic Processes.

Electrolytic Production of White Lead.—C. P. Townsend, 847,032 and 846,526, March 12, 1907. Application filed April 18, 1902. Patent 846,526 refers to the apparatus, 847,032 to the process.

In former attempts to electrolytically produce white lead it has been found that the product is irregular in composition and quality and deficient in covering power. These disadvantages may be overcome according to the inventor if a definite circulation of the electrolyte is produced successively past the respective electrodes and preferably in the direction from the anode past or through the cathode. Fig. 3 shows one type of apparatus. B is the anode which consists of lead, while the perforated plates C constitute the cathode. They may consist of perforated metal or of wire gauze. The electrolyte—which is a mixture of sodium nitrate or acetate with sodium carbonate in the approximate proportion of 10 to 1—passes through the apparatus along the paths indicated by the arrows. From the lateral channels *gg* the electrolyte enters into the outlets D and then passes through the pipe G into the filter press H. The filtered electrolyte flows into tank I, and is then raised again by means of the fan M into the electrolytic vat. The circulation is, of course, not confined to the electrolyte alone, but the pigment or insoluble anodic product detaches

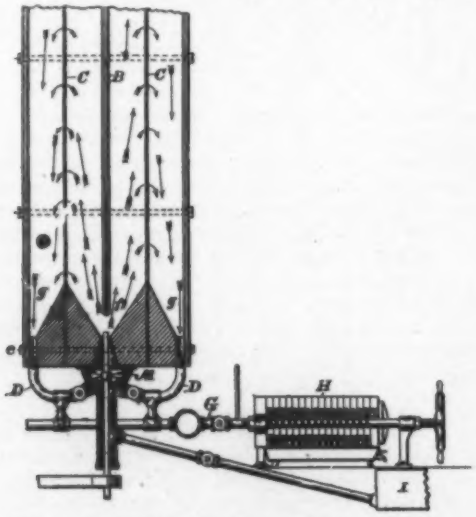


FIG. 3.—ELECTROLYTIC PRODUCTION OF WHITE LEAD.

itself from the anode B and is carried upwardly with the electrolyte through the cathode and then to the filter press. The area of the cathode is much smaller than that of the anode. "This is of great assistance in preventing the formation of lead sponge by reduction of the pigment, not only because the speed of circulation through the cathode is thereby increased and the pigment prevented from adhering to it, but because the electrolytic reduction itself proceeds more slowly when the cathode area is relatively small and the current density at the cathode is relatively high." The best current density depends to some extent on the speed of circulation of the electrolyte. With a higher current density a more rapid circulation may be employed. To regenerate the electrolyte it is, of course, necessary to add continuously or at intervals carbon dioxide and water in quantity equal to that removed by the pigment. The carbon dioxide may be made use of for effecting better circulation.

Reduction of Metallic Sulphides.—Edward L. Anderson, 846,642, March 12, 1907. Application filed Dec. 26,6 1905.

The object is the cathodic reduction of metals from their sulphides with the simultaneous development of hydrogen

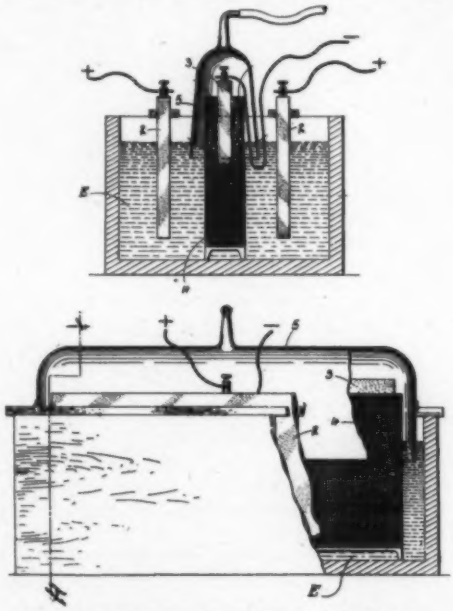


FIG. 4.—CATHODIC REDUCTION OF SULPHIDES.

sulphide. In its general scheme the process is, therefore, similar to the Salom lead reduction process (our Vol. I., p. 18). The upper diagram in Fig. 4 shows a vertical transverse section of the reduction tank, and the lower diagram is a part side elevation and part section of the same; 2 and 2 are carbon or graphite anodes, while the carbon or graphite plate 3 represents the cathode, which is surrounded by a wire basket 4 suitably laced or enameled, so as not to be effected by the electrolyte E, which is silico-hydrofluoric (hydrofluosilicic) acid, H_2SiF_6. By means of the bell 5 the H_2S gas is carried off to be burned in gas engines driving electric generators. For instance, to get zinc from zinc sulphide the ore is ground to about 15 or 20-mesh, and filled into the basket 4 around the cathode. The cathodic action is the formation of H_2S gas, which is carried off, while simultaneously the metallic zinc reduced from the sulphide goes into solution as $ZnSiF_6$. From this solution the zinc is then at once deposited on the cathode.

Insulating Diaphragm for Cells with Molten Electrolytes.—G. O. Seward and F. von Kuegelen, 842,256, Jan. 29, 1907. Application filed July 17, 1905.

It has formerly been proposed to form a non-conductive diaphragm or partition within a molten electrolyte by using simply a water-cooled iron partition. It was supposed that the layer of chilled salt which then forms on the surface of the iron curtain would prevent the current to pass through. It is, however, not possible to maintain such a layer of chilled electrolyte in sufficient thickness and at a sufficiently low temperature to avoid its conducting a portion of the current from the electrolyte to the curtain. The curtain will then act as an intermediate electrolyte and become inefficient as a diaphragm. To prevent this, the present inventors use two such water-cooled curtains whereby they are enabled to get a

sufficiently thick mass of electrolyte between the two curtains to act as insulating diaphragm. The construction is shown in Fig. 5. D is the cathode and A the anode. Between the

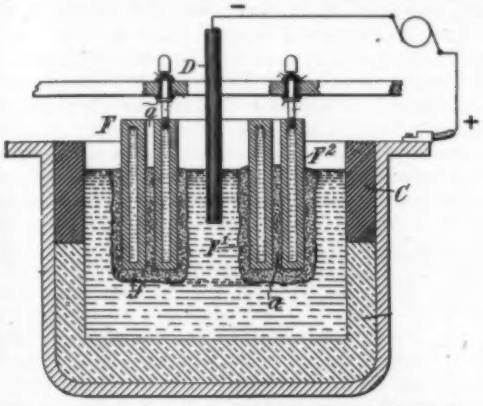


FIG. 5.—DIAPHRAGM CELL FOR MOLTEN ELECTROLYTES.

anode and cathode is introduced the non-conductive partition F, which is made of two tubular cylindrical water-cooled curtains F' and F², the former being arranged concentrically within the latter, leaving an annular space *a* of uniform width between them. The curtains F' F² may be made of cast iron cored out to form a water space or channel through which a circulation of water is maintained to cool them. The two curtains are necessarily insulated from one another. By the circulation of the water the hollow curtains are so cooled that the layer of molten salt is congealed upon them and the space *a* between the two curtains is filled up with congealed salt.

Precipitation of Gold from Cyanide.—W. A. Hendryx, 834,575, Oct. 30, 1906. Application filed Jan. 17, 1906.

Mr. Hendryx' work in the cyanide process has been repeatedly noticed in these columns, for instance, our Vol. IV., p. 504. The present patent relates to the construction of "precipitating cells." Such a cell consists of a rectangular wooden frame, the exterior of which is covered by a filtering canvas sack, while the inside contains graphite anodes and copper, tin or iron cathodes. A number of such precipitating cells are placed in an extraction tank and submerged in the body of ore-pulp of metal-bearing cyanide solution contained therein. The interiors of the precipitating cells are connected with the suction-line of a pump, whereby the solution is carried into the cells, while the suspended matter is retained outside by means of the filter cloth. The gold is electrolytically deposited on the cathodes. At suitable intervals the pulp or slime adhering to the outside of the filtering canvas is detached by forcing air into the inside of the cells.

Electroplating Apparatus.—D. F. Broderick, 839,719, Dec. 25, 1906. Application filed July 27, 1906.

Mechanical details of construction of dipping mechanism, whereby the articles to be plated and carried by a work-holder, are automatically dipped successively into the various plating and washing baths. An endless carrier is provided for the work-holders and so associated "with a series of tanks and work-holder-elevating devices that the endless carrier can be utilized throughout its full extent of travel for moving the work-holders and causing the successive immersing of the work or articles to be plated into baths arranged under the carrier, whereby the complete process of plating and washing can be accomplished while the article carried by a holder is moving from a given starting point until it returns to such starting point."

Recovery of Metals from Fumes.—L. Dion, 840,480, Jan. 8, 1907. Application filed March 4, 1904.

The inventor claims he can concentrate and recover metals contained in the fumes and vapors arising from molten metal and can simultaneously segregate them from one another by the following method. The fumes are first subjected to the action of an electric current supplied through appropriate electrodes in a dry chamber, whereby particles of metals are concentrated. The fumes, with the particles thus concentrated, are then subjected to the action of an electric current in a liquid bath, "by which means not only are the particles or masses already precipitated by the action of the electric current in the dry chamber aggregated into larger masses, but any particles of metal remaining to the fumes, gases, or vapors after passing through that chamber also precipitated and recovered."

BATTERIES.

Metal for Storage Battery Grids.—W. Morrison, 842,801, Jan. 29. Application filed Jan. 20, 1905.

To make a strong solid grid, free from pores, the inventor uses a lead-antimony-phosphorus alloy. To prepare this, he mixes an alloy of 100 pounds of lead and 12 pounds of antimony, while in molten condition, with 2 ounces of phosphorus. "The phosphorus must be added in small portions first after the molten metal is stirred and then the air-tight cover of the pot is put on." This is repeated until the desired amount is added. Two other methods are also described.

Storage Battery.—S. Lake, 844,815, Feb. 19, 1907. Application filed May 26, 1904.

In storage batteries of large capacities it is important to have the walls of the cells stiff enough to prevent them from bulging outwardly into contact with those of the adjacent cells; to insure against leakage of electrolyte from any cause and thus avoid liability to short-circuiting. How the inventor endeavors to attain this end is indicated by the first claim: "A battery cell comprising an inner tank and an outer tank inclosing the same, rigid plates of material infusible at moderate temperatures interposed between and spaced from the walls of said tanks, and a filling of readily-fusible waterproof insulating material also interposed between the walls of said tanks and occupying the spaces intermediate said plates and the walls of said tanks."

Storage Battery Plate.—Joseph Bijur, 845,048, Feb. 26, 1907. Application filed Aug. 14, 1906. (Assigned to General Storage Battery Co.)

Details of construction of a paneled plate, the features of which will become clear from the description of a single panel. Within its rectangular frame are placed horizontally a large number of alternately straight, flat strips and of corrugated strips so as to fill the whole interior of the frame. The surfaces of both the straight and the corrugated strips form the active surface which is therefore large. To keep all these strips in proper position, they are provided with holes, and rods are cast through these holes from the frame.

Negative Plate.—Joseph Bijur, 845,391, Feb. 26, 1907. Application filed April 8, 1906. (Assigned to General Storage Battery Co.)

In the long series of storage battery patents of Mr. Joseph Bijur, president of the General Storage Battery Co., of New York, with works at Boonton, N. J., this patent is a particularly important one, since it solves the old difficulty of the loss of capacity in negatives, which has been the bane of manufacturers in the past. It is reported that this method has been practiced by the General Storage Battery Co. for over a year with full success. The loss of capacity in the plate is prevented by impregnating the plate with an inert material, not chemically acted upon by the electrolyte, like carbon. A peculiar and important feature is that the plates are impregnated after they have been formed, so that the process is applicable to plates which have been in use. The process is applicable to Plante plates and pasted plates. In the case of Plante plates the plate is first formed and reduced to a spongy lead in the

usual manner and allowed to dry. It is then soaked for about 10 minutes in a solution of sugar of about 2 to 30 per cent. The solution penetrates readily the pores of the plate at ordinary temperatures. The solution is then rinsed from the surface of the plate and the plate is dried by permitting it to stand in air. If the plate is all pure lead, it is then baked in any suitable apparatus at a temperature of from 240° to 300° C. for 5 or 10 minutes, until the sugar is completely carbonized. If the plate has alloy parts, as, for instance, an antimonious lead frame (with a lower melting point than pure lead), the plate is heated in such a manner that the frame is subjected to less heat than the grills. There is, of course, no difficulty in doing this, and a suitable heating apparatus is described in the patent. In the case of pasted plates, the finished dried negative plate is placed in the sugar solution for about 10 minutes. It is then taken out, dried in air and heated in the same way as in the case of the Plante plate, whereby the organic material in the pores is again carbonized. The carbon may thus be introduced into the pores of the active mass of pasted plates even after they have been in use and have partially lost their capacity from the lack of such material, thereby again raising the capacity of the plate. "The invention also contemplates soaking a Plante or pasted plate in a solution of suitable material—as, for instance, sodium silicate—in which case the inactive material is deposited in the pores of the active mass in the plate chemically instead of by drying and carbonization." Plante plates treated by this process have a light brownish-gray color instead of the usual lead-gray color of the Plante plate. They have a very porous and spongy appearance, as distinguished from ordinary Plante plates, not treated by this process. In use the capacity of the plate either remains substantially the same or rises instead of falling off.

Battery Jar.—J. Y. Bradbury, 842,616, Jan. 29, 1907. Application filed Oct. 25, 1906.

To get a battery jar which can be handled roughly without danger, he uses a cell of hard rubber and provides it with a jacket of flexible fibrous material, such as cloth which is rendered impervious to liquids and is acid and alkali-proof. This external armor covers the bottom and the greater portion of the sides and ends of the cell, but leaves the upper portion of the cell uncovered so that when a number of armored cells are placed in a retaining box, an air space is formed between the upper ends of the contiguous cells. The object is to prevent leakage from one cell to the next, due to moisture.

Storage Battery Switch.—S. A. Leonard, 842,405, Jan. 29, 1907. Application filed May 14, 1906.

Mechanical details of construction of switches for storage batteries. The first claim reads: "In switches for storage batteries a suitable supporting frame, longitudinally movable plates mounted oppositely on said frame, and brush and plate connections for the current operatively related to said plates."

Primary Battery.—F. J. Decker, 842,389, Jan. 29, 1907. Application filed Feb. 27, 1904.

This is another patent relating to the Decker battery (our Vol. IV., p. 441, and our Vol. V., p. 58 and 104). The first claim relates to "a battery comprising a plurality of cells, a duct extending into the interior of each of said cells, a conduit communicating with a plurality of said ducts, and means for connecting said conduit to said ducts, said connection permitting the removal of said cells and ducts without moving said conduit."

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

INDUSTRIAL ELECTROCHEMISTRY.

Electrolytic Precipitation of Gold from Cyanide Solutions.—In our Vol. II., pp. 55, 131 and 215, we gave detailed information on the Siemens-Halske electrolytic process for precipitating gold from cyanide, as operated in South Africa and America, and on modifications of this process due to Mr. Charles Butters. In a note by Mr. Butters in our Vol. II., p. 207, he then brought out that in his plants he now uses a combined zinc and electrolytic precipitation process. An interesting article by Mr. Charles B. Richmond, in the *Engineering and Mining Journal* of March 16, describes the process as it is now operated at the (Butters) San Sebastian Mine in San Salvador. Very small amounts of silver are present in the ore, which is a complex mixture of sulphides. Free gold, in the unoxidized zone, is of common occurrence, but the greater part is combined with the telluride and sulphide. The quantity of copper varies. The cycle of the ore from the time it reaches the mill is as follows: Breaking in Blake-Marsden crushers, drying in rotary dryer crushing in ball mills to 40-mesh, roasting in Jackling furnaces, whereby the sulphur is reduced from 3.5 to 0.5 per cent; elevation of the roasted ore with weak cyanide solution to sand collecting tanks, separation of sand and slime, agitation and filter box treatment for slime, gravity percolation for sand, recovery of gold and copper by means of combined electrical and zinc precipitation. The gold extraction has averaged 95 per cent during the period of operation, about four years, and the only drawback was that the product contained gold and copper precipitated on lead foil cathodes. Cupellation refining was impossible because of the large excess of copper, hence these cathodes were milled, sampled and sent to the smelters.

The average composition of the product was 3.37 per cent of gold, 65 per cent of copper, the remainder being lead. The smelters paid nothing for the copper and lead, and only 32s. 6d. per fine ounce for the gold. Tinned iron cathodes had been tried at this plant, but were discarded in favor of lead, for two reasons: first, because of the excessive quantity of gold necessary to coat the plates before any values would slime to the bottom; and, second, because of the difficulty and the loss in scraping off the dense hard deposit of gold and copper. The present author has now devised a process by which the gold, copper and lead are sufficiently separated to bring their respective values. The lead cathodes are returned to the precipitation boxes. The copper is shipped to the smelters as copper cement, and the gold precipitated is melted into bars, which are shipped to the refiners, and realize 84s. 11½d. per fine ounce, less a charge of 2d. per bullion ounce for refining.

The cyanide solutions first enter the electric precipitation box containing anodes of peroxidized lead plates ⅛ inch thick, and lead cathodes of 1/16 inch thick. The anodic current density is 1 amp. per square foot. The gold and copper precipitated on the cathodes form dense hard coatings. There is no tendency to slime. However, there is a gradual accumulation of low-grade precipitate on the anodes and on the bottom of the box with the gold content of from 5 to 50 ounces per ton. From the electric precipitation box the solution flows into the zinc boxes. The electric section extracts from 80 to 90 per cent of the gold, nearly all the copper that is precipitated, and re-

generates a high percentage of the cyanide. The tail solution from the zinc carries from 0.06 to 0.10 dwt. per ton, giving a total extraction of about 99.5 per cent. The special new feature is the treatment of the lead cathodes with their dense gold and copper deposit from the electric precipitation boxes. They are treated as anodes in acid boxes containing dilute (2 to 3 per cent) sulphuric acid. They hang in a wooden frame with closed bottom and open sides; over each is stretched a cotton cloth sack. The cathodes are lead plates. As the result of electrolysis the copper on the now anode plates dissolves, passes through the cloth of the frame and precipitates on the cathodes, where it slimes and falls to the bottom of the box. The gold released from the copper falls to the bottom of the anode frame. In this way the gold and copper are obtained separately, while the lead plates are brought back to their original condition and are used over again as cathodes in the cyanide precipitation box.

Electrolytic Rectifier.—*L'Industrie Electrique*, of Feb. 25, contains a note on a paper by de Faria, presented before the (French) Physical Society, on his electrolytic rectifier, which does not differ in principle from the old type. The special feature seems to be an automatic regulation of the electrolyte, which prevents the temperature from rising to a dangerous value. The electrodes consist of pure commercial aluminium and antimonial lead, and the electrolyte is a solution of sodium phosphate.

Electroplating.—In the February issue of the *Journal* of Société Belge d'Elec. and in the issue of Feb. 25 of *L'Industrie Electrique*, R. Goldschmidt describes a useful method of electroplating large articles which cannot be conveniently placed in an electrolytic tank. He soaks an ordinary brush with the electrolyte and connects it with the positive pole by means of a wire of the metal to be deposited wound around the top of the brush. The article to be electroplated is made the negative pole. When the article is touched with the brush the circuit is closed and the metal is deposited on the article. With a 110-volt supply circuit he uses in series with the brush six lamps of 50 c.p., which represent a resistance of about 750 ohms. The current thus obtained is about 1/10 amp. The following solutions are recommended: For silver plating, use 20 grams of silver nitrate, 30 grams of potassium cyanide, 0.4 gram of ammonia (95 per cent), 2 grams of potassium formate, and add water to get one liter of solution. For gold plating use 6 grams of gold chloride, 2 grams of ammonia (96 per cent), 13 grams of potassium cyanide and add water to get one liter solution; finally add 2 grams of potassium formate. For copper plating use 180 grams copper sulphate, 60 grams concentrated sulphuric acid and add water to get one liter of solution; then add 10 grams of alcohol (96 per cent). For nickel plating use 60 grams of nickel sulphate, 20 grams of sodium sulphate, 20 grams of sodium citrate and add water to get one liter of solution. For nickel plating it is preferable to use only a low tension, 2 or 4 volts.

Fuels.

Coal.—Goutal, in the *Journal für Gasbeleuchtung*, 1905, Vol. 48, p. 1,007, has given a reasonable solution of the very interesting question of calculating the calorific power of a coal directly from its proximate analysis,—fixed carbon, volatile matter, water, ash. The percentage of volatile matter is to be recalculated in per cent of the weight of fixed carbon plus volatile matter (pure fuel), and then the calorific power calculated by giving the fixed carbon present in the fuel a calorific power of 8,200, and the volatile matter present a calorific power which varies with the calculated per cent of volatile matter in the pure fuel, as follows: (Multiply figure given by 100.)

Per cent volatile matter.....	5	6	7	8	9	10	11	12	13
Calorific power (÷ 100).....	145	147	139	136	133	130	127	124	122
Per cent volatile matter.....	14	15	16	17	18	19	20	21	22
Calorific power (÷ 100).....	120	117	115	113	112	110	109	108	107
Per cent volatile matter.....	23	24	25	26	27	28	29	30	31
Calorific power (÷ 100).....	105	104	103	102	101	100	99	98	97
Per cent volatile matter.....	32	33	34	35	36	37	38	39	40
Calorific power (÷ 100).....	97	96	95	94	91	88	85	82	80

Example: A coal contained 86.70 per cent fixed carbon, and 10.05 per cent volatile combustible matter (besides 1.45 per cent ash and 1.80 per cent H^2O). It contained, therefore, 96.75 per cent of pure fuel, of which $10.05 \div 96.75 = 0.104 =$ 10.4 per cent is volatile matter.

The calorific power of the volatile matter corresponding to 10.4 per cent in the coal is therefore, from the table $129 \times 100 = 12{,}900$ Calories. The calorific power of the coal is therefore:

			Cal.
Carbon	0.8670 ×	8,200 =	....
Vol. matter	0.1005 ×	12,900 =	....
		Sum =	8,406
Direct experiment			8,404

It is stated that the calculation is within 1 per cent of the values obtained calorimetrically. It should be noted that these values are the higher ones, for coal burned to vapor of water condensed, and that a correction of 50 to 350 Calories would be needed to get the lower or practical calorific power.

Utilization of Peat.—The *Elek. Zeitschrift* of March 7 contains an interesting review of the attempts which have been made to utilize the extended deposits of peat in Germany. In view of the absence of large water powers this problem is of special interest to Germany, but may become of industrial importance also in this country. The peat industry has really lost ground somewhat in Germany during the last ten or twenty years. Of the methods for coking peat that of Ziegler has been found most successful. It yields a very strong and pure coke and, as byproducts, a certain proportion of the nitrogen in the peat in form of ammonia and several other valuable products of dry distillation. But although the Ziegler process has been technically as well as financially successful on the Upper Bavarian coke works in Bauerberg, it would not seem to be suitable for very large plants. The writer believes that a really rational utilization of peat on a large scale is possible only by perfect gasification of the fuel in large gas producers in connection with large gas engines driving electric generators. Prof. Frank (of cyanamide fame) has recently delivered a lecture on the use of blast furnace and coke oven gases for the operation of gas engines. Up to April 1, 1906, there were in German mines and metallurgical works 391 gas engine plants, with a total capacity of 416,000 hp., the largest capacity of a single gas engine being 5,000 hp. Reference is then made to the well-known Mond process, which yields producer gas and ammonium sulphate as a by-product. Caro has improved this process in so far as he gasifies poor fuel in a mixture of air and over-heated steam. Extended experiments with this modified process on the Mond works in Stockton have shown that it is possible to treat directly wet peat, containing 50 to 55 per cent of water, with a simultaneous increase of output of ammonium sulphate. On the bituminous coal mine "Mont Cenis," near Herne, in Germany, a new large gas producer plant is to be erected in which the new process is to be used for treating waste coal and peat. The chief result of the success of this first large undertaking would be to render available the use of wet non-briquetted peat in gas producers, while the ammonium sulphate obtained as a by-product will assure in itself a fair interest on the capital invested. The latter is $187,500.

Gold.

Loss of Gold in Placer Mining.—Mr. Dennis Storall, in *Min. and Scientific Press*, Feb. 23, calls attention to the fact that in order to catch as much as possible of the gold and platinum in placer mining operations, the slime boxes must be provided with more efficient riffles than those that are used generally. He maintains that the riffle which saves the greatest percentage is the riffle that offers the least resistance, but which allows the gold and the platinum sand ample chance to settle. The block riffle, which is used extensively and consists of 5 or

6-inch blocks sawed, squared and set in pairs on the sluice floor, the blocks being cut and set with the lower side higher than the upper, so as to increase agitation and resistance, is a "boiling" riffle of the worst kind. About half the gold and all the platinum, kept constantly on the surface of the current by the agitation of the water, fails to settle and flows over the dump. An efficient riffle is made of railroad rails placed bottom-side up, tilted just a little and set an inch or more apart on the sluice floor. Such a riffle gives the gold ample opportunity to settle in the crevices between the rails. According to the author a cheaper and more easily handled riffle is made by setting 2 x 4-inch scantlings edgewise on the sluice floor and 2 inches apart. The top edge is slightly leveled and strap iron is bolted on it, the straps being half an inch wider than the scantling. This riffle does not produce agitation, and the gold and platinum sand settle in the crevices between the scantling. Such riffles, cleated together and made of a width and length to fit each sluice section, can be readily set in the boxes and easily lifted out and raised at clean-up time.

Cyanidation of Raw Pyritic Concentrates.—The ore of the mines of the Socorro Gold Co., Yuma County, Arizona, consists of quartz with pyrite as the principal mineralizer, with occasional galena and corellite. It is treated in a 20-stamp mill. No provision was made to treat the concentrates, and they could not be shipped on account of the high freight and smelter charges. Attempts were therefore made to cyanide them raw locally, and the practice finally adopted is described by Mr. F. Smith in the *Bi-Monthly Bulletin* of the Am. Inst. of Mining Engineers, January. In laboratory experiments it was found that if the concentrates were ground to 100-mesh and agitated with cyanide for 32 hours, about 85 per cent of the gold and 70 per cent of the silver could be extracted, with an expense of about 6.5 pounds of cyanide per ton. The mill practice, however, has given much beter results. The yield of concentrates in the mill is about 2 tons per day. The treatment is carried out in a 5-foot old-style amalgamating pan with wooden sides. The pan is charged with about 1 ton of solution, carrying 6 pounds of cyanide, 6 pounds of lime and 1.5 tons of raw concentrates. The pan runs at about 75 r. p. m., and grinding is continued for 8½ hours, about 2 pounds more of cyanide being added during the day as the strength fails. At the close of the grinding the temperature of the mass has risen about 40° F. above the outside temperature, a fact to which the author attributes the better extraction of the values in mill practice as compared with laboratory results. When the grinding is finished the material is discharged into one of two 15-ton leaching tanks. When a charge from the grinder has been put into the leaching tank and the solution has settled to some extent the material in the tank is covered with dry middlings from the table in order to facilitate percolation. When a leaching tank is filled it is continuously leached with cyanide solution, while the other tank is being filled, and one tank receives about four days of extra leaching after it is filled and three days washing before discharge. The average extraction in a six months' run has been about 94 per cent of the total values; the whole plant is operated, without extra cost, by the man who attends the tables and vanners. The total cyanide consumption is about 8 pounds per ton. The total cost of treating the concentrates, including the values left in the tailings, does not materially exceed $5.00 per ton.

Grinding in Tube Mills at the Waihi Gold Mine, Waihi, New Zealand.—The Waihi mine, the largest in New Zealand, was one of the pioneers in the introduction of the tube mills into its metallurgical practice, and has found them so effective that it has been decided to equip the Victoria mill of 200 stamps with a plant of at least nine tube mills. The Waihi mill has three mills of the Davidsen 22-foot type, running at 27.5 r. p. m. and carrying a load of 5.5 tons of flints. The quantity of the later consumed per week is 18 cwt. As in every mill where tube mills are installed, experiments have been conducted with various kinds of linings. The lining now adopted is the so-called "honeycomb" lining of H. P. Barry, the construction of which has been described in the Synoposis. As used at the Waihi plant, it consists of a light cast iron frame, 22 x 14 x 3 inches deep, shaped to the curve of the mill and divided into four or six compartments by thin walls. A temporary sheet iron back is fastened to the frame, and each compartment is then firmly packed with rough lumps of hard quartz or quartzite, varying in size up to 4 inches square. These lumps are bedded in with a mixture of Portland cement, coarse and fine sand. The linings thus formed are allowed to set for several weeks, preferably under exhaust steam, the longer the better, before they are being placed in the mill. This method of lining the mill requires a much shorter stoppage for relining than was necessary when the lining was made with quartzite blocks. The frames fit neatly with each other and with the shell of the mill, and, therefore, only a small quantity of cementing material is required. The grinding efficiency is stated to be quite equal to that of the quartzite blocks and the lining, including labor, costs about $175, as compared with $400 for lining with quartzite blocks. The author states that in addition to the benefits of an increased tonnage, due to the substitution of a 20-mesh screen in place of a 40-mesh in the batteries, the tube mills have favorably influenced the extraction, as before their use the combined sand and slime residues assayed 31 grains of gold per ton, or an extraction of 89.8 per cent, whereas with the tube mill plant the combined residues assay 24 grains of gold per ton, representing an extraction of over 92 per cent. The effect on the slimes treatment has also been very beneficial, as a large proportion of sand is ground so fine by the mills that it passes the spitzlutte with the slimes, and thus makes the latter more easily treatable either by the filter presses or the vacuum process. The cost of running the tube mills per ton of sand passed through is as follows: Power, 12.5 cents; flints and liners, 14.0 cents; labor, repairs and stores, 1.5 cents; a total of 28.0 cents, or 18.2 cents per ton of ore crushed on the total mill tonnage.

The Tavener Process.—In *Bulletin* No. 29 of the Institution of Mining and Metallurgy, London, Feb. 14, Mr. L. Swinney summarizes the advantages of the Tavener process over the old process of pot smelting in the recovery of gold from the by-products of the extractor house as follows: (1) In the Tavener process about twenty-one pots of slag, more or less, are obtained from the by-products smelt per month, running about 8 dwt. per ton and weighing about 1.5 tons. This material is taken to the waste dump and its value, about 12 dwt. of gold, or $12.00, is lost per month beyond recovery. On the other hand, in the pot smelting process about 1 ton of slag is obtained which is crushed up and panned for the metallics which it contains, but the value of which even after panning is about 20 ounces per ton. By the addition of 5 ounces of gold contained in the pots and liners and 5 ounces for sundries the total value is brought up to 30 ounces per ton in the rejected products. This slag is stored up and cannot be treated further under the present methods of working. If this 1 ton of slag of the value of 30 ounces were capable of being sold, the buyer would only pay 50s. an ounce, thus entailing a dead loss of 15s. per ounce, or a total of £22 10s. per month. (2) The gold slime is merely dried in the drying furnace in the Tavener process, and is slightly moist when mixed up with the fluxes, whereas in the pot process it is calcined. Therefore, practically no dust can be carried up the flue by the draught, in the former process, and the loss is slight when compared with the amount lost in the pot-smelting process. (3) At any time the whole amount of gold in slags, furnace sweepings, cupels, etc., could be recovered by the smelter at 40 hours notice by means of the by-products smelt. Thus there is no gold locked up in any of the by-products and very little is sent to the dump. In the pot process, on the contrary, a considerable amount of gold is locked up in the slags, liners, etc., of which gold only about half the value can be saved after

many months waiting for a sufficient accumulation of slags to pay the buyer to take it away. (4) During the smelting process the bath of lead contains at the most 10 per cent of gold, this dilution of the charge rendering the working much more safe than that of the pot-smelting process. The loss of gold in case any lead is lost is therefore small, and at any time the lead can be treated again. On the other hand, in pot smelting the smelter works with a rich product and there is danger of some of the pots breaking and scattering their gold contents. (5). In the pot process the manganese used drives the silver into the slag, the resulting bullion being often 900 fine, but containing very little silver. The latter can be obtained by a clean-up smelt, but the resulting bullion is so rich in silver that an addition of gold-bearing bullion has to be made so as to bring it up to the standard of fineness required by the buyers.

The Assay of Silver Bullion.—Mr. J. Clennell, in the *Bulletin* of the Institution of Mining and Metallurgy, London, Feb. 14, gives a description of a modification of Volhard's method used for preliminary bullion assays at the Redjang Lebong mine, Sumatra. The principal standard solution was nitrate of silver made by dissolving 2.5 gr. pure silver foil in dilute nitric acid, helping to expel nitrous fumes and diluting to one liter. A solution of potassium or ammonium thiocyanate was prepared and adjusted to correspond exactly with this silver nitrate solution. This particular strength of silver nitrate and thiocyanate gave a convenient volume, namely, about 30 cc. of thiocyanate used, in titrating the solution from 0.1 gm. of bullion. The latter amount was weighed out, dissolved in about 50 cc. of nitric acid (25 per cent by vol.), the liquid, after boiling well, decanted into a somewhat large flank and the residue boiled again with stronger acid (75 per cent by vol.). The residue was washed, dried, ignited and weighed as gold. The solution was cooled under the tap, mixed with about 5 cc. of a ferric indicator, made by oxidizing a fairly strong ferrous sulphate solution, and a measured excess of the thiocyanate added with thorough agitation. The red color was then removed by the careful addition, drop by drop, of the standard silver nitrate solution. The finishing point was fairly sharp, it being possible to estimate with care to about 0.05 cc., corresponding to 1.25 parts of silver per 1,000. When it was sought to increase the accuracy by using a weaker silver nitrate solution for titrating back, or a weaker thiocyanate solution for restoring the color, it was found that the exact finishing point was too indefinite to be of any practical value. Increasing the amount of bullion was inconvenient, owing to the bulk of the precipitate, but sometimes 0.5 gm. of bullion were treated as described above, slightly increasing the amount of acid. After adding the excess of this cyanate the flask was well agitated and allowed to stand until the precipitate had thoroughly settled. It was then decanted and washed once or twice by agitation and settling, no filter being used. The liquid poured off was then titrated back with silver nitrate and the accuracy could thus be increased to 0.25 parts silver per 1,000. The method was never relied on as final, but as a useful indication as a preliminary to the fire assay, the results being generally somewhat lower than the latter; copper, lead, zinc and selenium in ordinary amounts did not interfere.

Roasting Argentiferous Cobalt-Nickel Arsenides.—The results of an investigation of the behavior of the argentiferous cobalt-nickel arsenides of the Temiskaming (Ont.) district during washing, made in the laboratory of Columbia University by Prof. H. M. Howe, Dr. W. Campbell and Mr. C. Knight, are given in a paper published in the *Bi-Monthly Bulletin* of the Am. Inst. of Mining Engineers, January. The ores of the district are native silver, with small amounts of dyscrasite (Ag_3Sb), argentite (Ag_2S), pyrargyrite (Ag_3SbS_3), smaltite ($CoAs_3$), chloanthite ($NiAs_3$) and niccolite (NiAs), while mispickel (FeAsS) and cobaltite (CoAsS) occur in similar quantities. The ore used in the investigation was chiefly smaltite, containing a total amount of 689 ounces silver to the ton and 56 per cent of arsenic. The roasting was done by heating the ore, which was held in shallow pans in a gas furnace. The object of the investigation was to ascertain (1) the temperature at which the arsenic is most rapidly expelled; (2) the thoroughness with which it is expelled by prolonged roasting at this temperature and (3) the effect of adding charcoal near the end or at the beginning of the roast. It was found that 15 per cent of arsenic per 100 of ore, that is, 27 per cent of the total arsenic is expelled below 700°, but that the rest of the arsenic is not expelled until the temperature reaches about 840°, when rapid expulsion again sets in. By revolving at temperatures above 840°, the percentage of arsenic can be further reduced by about 24 per cent, that is, down to 17 per cent in the ore, from the original 56 per cent; in this range of temperature the arsenic is removed much faster than at lower temperatures. On account of this behavior the authors conclude that the behavior of smaltite during washing is probably analogous to that of pyrite, which latter loses its first atom of sulphur much more readily than the second, though unlike the pyrite, which frits when suddenly heated, the smaltite ore may be raised suddenly to 800° without harm, as it remains open and porous. The addition of charcoal either at the beginning or towards the end of the roast failed to increase the expulsion of arsenic. Finer grinding of the ore, after it had been roasted once, and reroasting at about 880° C., showed no further expulsion of arsenic, due to fine grinding.

Copper.

Magnesium for Copper Sand Castings.—The difficulty of producing sound copper castings and its remedies are discussed by Erwin S. Sperry in the *Brass World*, February. The well-known reason of the difficulty is that copper oxidizes very rapidly when melted, and the oxide dissolves in the molten copper as soon as formed. When the oxide is present in considerable amount, the copper runs sluggish and thick, and the castings are exceedingly porous. Charcoal, when used as covering for the copper, reduces some of the oxide, but not all. The most satisfactory remedy is the addition of a small quantity of a very strong reducing agent, which not only reduces the oxide but absorbs the gases. For sand casting of steel aluminium is most suitable, while for copper castings silicon (in the form of the alloy copper-silicon) has proven very satisfactory for many years. Sperry discusses the special advantages of magnesium instead of silicon for copper castings. All that is required in practice is to melt the copper and introduce a small piece of magnesium stick (about 2 ounces to 100 pounds of copper) into the metal, stir, allow the magnesium time to act and then pour into sand castings. The difficulty with using pure magnesium is that it is very light and tends to float on the surface of the copper and burn off without having any beneficial effect. It is, therefore, necessary to push it down under the surface with a pair of tongs. For this reason it is preferable not to use the very light pure magnesium, but a heavier magnesium—copper alloy, containing 90 per cent of copper and 10 per cent of magnesium. This alloy is prepared as follows: Melt 45 pounds of Lake copper under a good layer of charcoal, so that it will not oxidize any more than is necessary, and then introduce, one stick at a time, 5 pounds of stick-magnesium. The sticks occur in commerce in pieces about 6 inches long, and they are held with tongs and pushed down under the surface of the molten copper. When all the sticks have thus been introduced, the whole is stirred with a plumbago stirrer and then poured out into small strip ingots, so that small pieces may be easily obtained by cutting up. If 2 ounces of pure magnesium is the proper amount to be added to 100 pounds copper, then 20 ounces of this 10 per cent alloy should be used instead of the 2 ounces of pure magnesium. A special advantage of the magnesium is its high electrical conductivity; its addition to copper, therefore, does not impair the conductivity of the latter. But to get good copper castings for electrical purposes the best purest copper must be used; the magnesium will not remove arsenic, antimony, bismuth, tel-

lurium, iron and other impurities. Besides reducing the oxide in the copper the magnesium acts as an absorber of gases, especially carbon monoxide gas. The reaction yields carbon and magnesium oxide, that is, two solids from the gas, so that the possibility of the formation of blow-holes is removed.

The Constitution of Ferro-Cuprous Sulphides.—An extended laboratory investigation, undertaken for the purpose of ascertaining whether, as maintained by Gibb and Philp, ferrous and cuprous sulphides form the chemical compound $5Cu_2S$, FeS, was carried out by H. O. Hofman, W. Caypless and E. Harrington at the Massachusetts Institute of Technology. The author constructed the freezing point curve of the series ferrous sulphide cuprous sulphide, and subjected the various alloys to microscopical examination. The two sulphides were prepared by heating metallic iron and copper with stick sulphur. The results of the investigation, together with several microphotographs are given by the authors in the *Bi-Monthly Bulletin* of the Am. Inst. of Mining Engineers, January. They arrive at the following conclusions: 1. Ferrous and cuprous sulphides form no chemical but a eutectiferrous compound. 2. The structure of the eutectic of ferrous and cuprous sulphides becomes merged in that of the pure ferrous sulphide and cannot be distinguished from it. 3. The limited reciprocal solubility of the two sulphides diminishes along the cuprous sulphide branch of the curve, slowly at first, then more quickly; solubility is complete beyond the alloy FeS, 20 per cent, and Cu_2S, 80 per cent.

Zinc.

Decomposition of Blende.—The pyrite deposits of Meggen in Westphalia are of great importance to Germany for the manufacture of sulphuric acid. The residues, however, contain several per cent of zinc, as sulphide, and for that reason are not salable to the blast furnaces. It would be of considerable value if a process were found whereby the zinc and sulphur could be practically removed. C. A. Graumann undertook the study of this question in the Institute for Metallurgy and Electrometallurgy at Aachen, and reports his results in *Metallurgie* for Feb. 8. If this residue is heated to melting, it gives off some sulphur and some zinc vapors, but the amount thus expelled is insufficient. Mixtures of Fe^3O^3 and ZnS were heated in a magnesia-lined porcelain crucible in an electric resistance furnace, arranged for the passing of gases through the furnace. A mixture in the proportions $2Fe^3O^3$ to ZnS (16 and 4.88 grams, respectively), heated 1 hour to 1,280° in a current of nitrogen,acting as an indifferent gas. Only 10 per cent of the zinc and 20 to 30 per cent of the sulphur was removed in this way. The experiment was repeated in an atmosphere of air, which partly formed CO^2 with the carbon terminals used in the furnace. Held at 1,280° to 1,300° for an hour, 70 per cent of the zinc and 75 per cent of the sulphur were removed. Further experiments showed that ZnS heated with FeO gives off neither zinc nor sulphur. Mixtures of $ZnS + 3Fe^3O^3 \times 9C$ were then heated, the product at 1,200° was not melted, but sintered. Two experiments showed 87 and 93 per cent of the zinc and 42 and 60 per cent of the sulphur, respectively, eliminated. This result leads to the proposal of the following commercial process of treating blende itself. The blende is intimately mixed with finely divided iron, produced as sponge by the reduction of iron ore. The mixture is to be heated to 1,300°, when zinc distils off quantitatively

RECENT METALLURGICAL PATENTS.

Iron and Steel.

Roasting and Agglomerating Pyrite-Cinder.—Pyrite cinder and other pulverulent materials, on account of their dust-like nature, cannot be treated in the same manner as lumpy ore. It has been proposed (see, for instance, our Vol. IV., p. 351) to subject the pyrites cinder to a desulphurizing and agglomerating treatment in an inclined rotary kiln, whereby they are formed into coherent lumps or nodules. Serious difficulties have been encountered in this treatment on

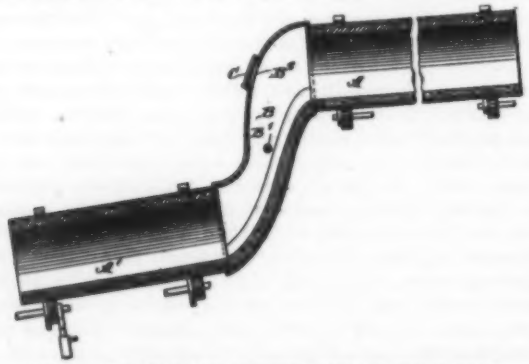


FIG. 1.—ROASTING AND AGGLOMERATING PYRITE CINDER.

account of the tendency of the pasty or semi-fused mass to adhere mechanically to the lining of the kiln or to enter into chemical combination with the kiln lining. R. F. Hill (847,664, March 19, 1907, assigned to General Chemical Co.) tries to overcome this difficulty by dividing the process into two steps. There are two rotary kilns A and A′, both inclined and one arranged at a higher level than the other. The upper kiln is considerably longer than the lower kiln, both are connected by any suitable pipe B, which is pivoted at B′, and is provided with a valve C, controlling an air inlet B^2. The upper kiln has an ordinary firebrick lining, while the lining of the lower kiln is basic, or at least neutral, being made of chrome, magnesite, bauxite, carborundum or steel brick. The connecting pipe B, or at least the lower portion of it, has also a basic or neutral lining. The pyrite cinder is fed at the upper end of the upper kiln A, while the fuel (as a coal-dust flame) enters at the lower end of the lower kiln A′. The roasting is performed in the upper kiln, and the temperature should be sufficient for this purpose but not high enough to melt the sulphide or cause the ore to sinter into lumps. In the lower kiln the material is formed into coherent lumps or nodules, the size of which depends upon the inclination of the lower kiln and also on the speed with which it is rotated. In the upper kiln no accretions will be formed to the linings, because the temperature is not high enough. In the lower kiln chemical combination between the ore and the lining is prevented by using a basic or neutral lining. If some of the fused material might adhere mechanically to the lining of the lower kiln, the remedy is to increase the temperature in the same, so as to melt the adhering mass away. Of course, this must not result in any undue rise of the temperature in the upper kiln, and this is prevented by admitting a greater amount of air through C to the connecting pipe. The mechanisms for rotating the two kilns are independent of each other. If the method is a commercial success, it is a new example of the old metallurgical rule that it is wrong policy to try to perform two different operations in one and the same apparatus.

Special Steel for Gun Forgings.—A patent of J. Churchward (846,979, March 12, 1907) refers to the production of a nickel-titanium steel stated to be specially valuable for constructing gun forgings and other steel parts which have to resist high temperatures and great strains. A good working

proportion is stated to be 95 per cent of steel (containing from 0.1 to 0.6 per cent of carbon), 3 per cent of nickel, 0.5 per cent of manganese and 1.5 per cent of titanium. The addition of manganese is necessary, since without the same the titanium, coming in contact with the steel, would get chilled (on account of the high melting point of the titanium), and would not properly assimilate with the steel. The procedure for making the alloy is as follows: "The titanium is melted in a crucible at a temperature, say, of about 3,000° C., and the steel and nickel are melted in another crucible and heated up to a point just below the point of volatilization of that metal having the lowest volatilizing point. The manganese is now added to the molten titanium, and while it is being melted and incorporated with the latter the molten nickel steel alloy from the other crucible is gradually added by pouring it in, and the alloy formed may be poured or cast directly into the mold."

Special Steel for Electrical Purposes.—The indefatigable work of R. A. Hadfield, of Sheffield, England, in connection with special steels and kindred subjects is so well known that nothing more need be said. His latest American patent (842,403, Jan. 29, 1907) refers to an alloy having magnetic and electric properties especially adapting it for use in ballast coils, transformer plates and like electrical apparatus, for which it is of great importance to reduce the total magnetic and electric losses to the lowest possible degree. Besides pure iron (containing less than 0.12 per cent of carbon) the steel contains from 2 to 4.5 silicon, less than 0.7 manganese and less than 1.3 per cent aluminium. Two special prescriptions are as follows: The first is 0.08 C, 3.98 Si, 0.08 Mn, 0.95 Al.; the other is 0.09 C, 3.57 Si, 0.67 Mn and 0.1 Al.

PRECIOUS METALS.

Extraction of Gold from Refractory Ores.—A patent of J. W. Worsey and E. Hoal (846,768, March 12, 1907) describes a process primarily intended for extraction of gold from refractory ores, but stated to have a wider application. The ore is first crushed and screened and freed from aluminium silicates in any known manner. It is then "subjected to the action of nascent bromide of chlorine produced in the body and presence of the ore by mixing with the powdered or crushed ore or salt of chlorate of soda and bromide of soda, or a chlorate and bromide of other equivalent alkali or a chloride and bromide of an alkaline earth." The mixture is quietly stirred and some acid is added, and the temperature is raised from time to time until it rises to about 150° F. After this treatment has been carried out for about 4 hours, the temperature is raised to near boiling point for the purpose of completing the dissolving of the gold and finally to free it from excess of gaseous products. The whole is then allowed to settle; the clear solution is decanted away and the residue is washed with hot water. The solution and the wash water are mixed together, and any acid present is neutralized by means of a quantity of a solution of an alkali. A weak solution of a lead salt is then added and the whole vigorously stirred, and H_2S is added to the solution under proper stirring (by blowing air through) for 10 to 20 minutes. The whole then turns quite black, and in this condition the precipitated gold and lead sulphide settle out, and are collected, dried and the gold recovered by a smelting operation." To recover any residual gold contained in the ore previously separated from the gold solution, it is treated and washed in a vat with a weak solution of double or single ferroprussiate of ammonium and commercial soda carbonate after the manner of ordinary leaching. The effect of this is that all the residuary gold in the ore is dissolved out of it, and this is effected quickly—namely, within 24 hours. The gold is dissolved out in the first stage as bromo-chloride, and afterward when heated it resolves itself into auric chloride, $AuCl_3$. In this treatment the ferro-prussiate of ammonium acts as an expeditor or saver of time, and by it the residual ores of this and other gold-extraction processes can be economically treated and gold extracted and within a greatly reduced time. The solution resulting from this leaching operation can be treated like the previous solution and sulphide of hydrogen (H_2S) for the separation of the auriferous lead sulphide."

COPPER.

Chlorination Process.—O. Froelich (846,657, March 12, 1907) suggests the following wet process of treating copper ores and smelting products, for which comparatively high rapidity of operation, the absence of roasting and smelting and the possibility of treating the poorest ores and smelting products are claimed. The ore is first subjected in the absence of air to "a temperature between 150° and 800° C.," whereby the loose sulphur is driven off. The pyrite is changed by this operation in its chemical composition and can then be chlorinated much quicker and better. As chlorinating gases, chlorine, vapor of hydrochloric acid and of ferric chloride are used. The chlorine and the ferric chloride attack the sulphur compounds of copper, the hydrochloric acid and the ferric chloride attack the oxides of the copper. The proportion of the gases and vapors in the mixture are adjusted to the composition of the copper ore. Steam may also be added to the chlorinating mixture. It is preferable to make the temperature of the chlorination somewhat higher than the boiling point of the ferric chloride, about 300° C.; but it can be lower if the copper in the ore only occurs in combination with sulphur. In this case only chlorine gas is used. If iron is present in the ore it is chlorinated together with the copper, but more slowly, and the process is facilitated by a higher temperature. In order to regain the chlorine combined with the iron, the ore or smelting product is heated to about 300° or more after chlorination, and a certain quantity of air is introduced during this operation.

The ferric chloride is then evaporated, and by the air separated into oxide of iron and chlorine gas, both of which are collected. The chloride of copper is not changed by this operation. Then the ore is treated with hot water and the chloride of copper and, perhaps, a residue of the ferric chloride are extracted. The solution is then introduced into a rotating apparatus containing pieces of iron, and the metallic copper is deposited in form of cement copper. The solution now contains mainly ferrous chloride, and in order to regain the chlorine from it the solution is oxidized in a rotating drum to ferric chloride by means of an air blast. Then the water and the water of crystallization are driven off by heating, and by increasing the temperature over the boiling point of ferric chloride and introducing a certain quantity of air, chlorine gas and ferrous chloride are obtained. If, in this operation, hydrochloric acid is formed, it can be transformed into chlorine by treating with calcium hypochlorite.

Treatment of Copper Silicates.—To treat silicates of copper of vitreous and anhydrous formation by a leaching process, G. H. Waterbury (847,448, March 19, 1907) subjects them first to a light or red roast, by means of which the formation is quickly changed to a black oxide and the vitreous form of the silica is disintegrated. The ore thereby becomes amenable to the ordinary leaching process.

Roasting Process.—At another place of this department the description of a roasting process of A. Lotti for lead sulphide ore will be found, which is also suitable for the treatment of low-grade copper matte and of sulphide ores of copper, arsenical or antimonial. According to the inventor, a special advantage of his process is that there is no necessity of briquetting and that such ores are desulphurized and brought at the same time into excellent form for smelting. As in the case of lead, the ore is mixed with a proper proportion of molten slag, and in some cases a silicious sand or finely crushed fluorspar, rich in silica, is added prior to the application of the air blast. The air blast is, of course, applied while the sulphides are still at the temperature of combustion.

Flux for Welding.—T. W. Wilson (847,629, March 19, 1907), patents a flux consisting of approximately equal parts

of a borate, specifically borax, and a nitrate, specifically saltpetre. The components are finely divided and intimately mixed. The ends of the copper pieces to be welded are beveled, cleaned and heated and then the mixture of borax and saltpetre is applied and melted on the surface. The two copper ends are heated to whiteness and are then welded together by hammering or compression.

Lead.

Roasting Process.—The object of a patent of A. Lotti (847,017, March 12, 1907), is to use the waste heat of the smelting process (in form of hot slag) for the roasting of lead or copper sulphide ores and for agglomerating them at the same time into a form well adapted for treatment in the blast furnace. For this purpose lead sulphide ore is thrown into a predetermined quantity of molten slag and stirred rapidly, whereby a spongy incandescent material is produced, dense fumes of sulphur, sulphur dioxide and sulphur trioxide being evolved at the same time. The material thus produced consists of the sulphide and oxide of lead mixed with slag. This spongy and friable material, while being still at the temperature of combustion, is subjected to a compressed-air current and the remaining sulphur is thereby, to a large extent, burned off, the temperature of the charge rising considerably, even to bright red heat. After 1 or 3 hours the desulphurization is complete and the mass forms a single block of spongy character. It may be broken up to suitable size as preliminary to smelting in the blast furnace, or the block, while still red hot in its interior, may be passed on directly to the blast furnace. It sometimes happens in this process that ores, rich in sulphur, produce in the smelting a little more matte than if they were treated by the ordinary roasting process, but in such a case the difficulty may be overcome by retreating a certain proportion of the matte, crude or roasted, along with the ore. This has the advantage of oxidizing, to a large extent, the matte, which then acts as a flux in the subsequent smelting.

Nickel.

Treatment with Chlorine.—C. A. Diehl and W. Koehler (842,139, Jan. 22, 1907) have designed the following method for treating nickel ore, matte, etc., as a preliminary to leaching, for the purpose of getting a pure nickel chloride solution. The ore which may contain nickel, magnesium, aluminium, and iron, with or without gangue, is first chloridized by known methods in order to effect the more or less complete conversion of the several metals into chlorides. The chloridized ore is then heated in a drum in an atmosphere containing oxygen, either in form of free oxygen or air or steam. The temperature should be below the point of decomposition of nickel chloride, but above the point of decomposition of the remaining "base" metals. The reaction

$$M''Cl_2 + M_2'''Cl_6 + 2O_2 = M''O + M_2'''O_3 + 4Cl_2$$

takes place if the temperature is sufficiently high and air is used as oxidizing agent. On the other hand, if steam is used, the reaction is

$$M''Cl_2 + M_2'''Cl_6 + 4H_2O = M''O + M_2'''O_3 + 8HCl$$

If a mixture of air and steam is employed, the two reactions occur simultaneously, with the result that a mixture of chlorine and hydrochloric acid is obtained. These may be utilized for chloridizing further quantities of ore. In the above equations M'' represents, of course, a bivalent metal—magnesium, zinc, barium, etc.—and M''' a trivalent metal—iron, aluminium, etc. The chlorides of the trivalent metals are not easily completely decomposed by oxygen, but such decomposition occurs readily and with substantial completeness in presence of the chlorides or oxides of the divalent metals. For this reason if one or more divalent metals are not already present in the ore, they are preferably added. Under such conditions and at a suitable temperature, which, in the specific instance described, is below red heat, there occurs a combination between the oxides of the divalent and trivalent groups, forming double oxides of the general type $M''O_1M'''_2O_3$, which are relatively stable compounds and are only very slowly soluble in the water or dilute acid employed in the subsequent leaching. Their solubility is further decreased by heating in presence of hydrochloric acid or chlorine, as is the case in the practice of the method above described. Furthermore, the physical state of the resulting material is such as to permit the rapid and substantially complete extraction of the nickel by leaching with dilute acid. No injurious effect follows from the presence of an excess of divalent oxide, since any oxide or hydroxide of nickel which might be precipitated thereby is readily soluble in the dilute leaching acid.

Production of Nickel Chloride Solution.—R. W. E. MacIvor (846,492, March 12) patents the following process for treating nickel ores or oxidized nickel matte for the production of a solution of nickel chloride, from which the nickel may be recovered by known methods. The ground ore or oxide is "digested under pressure in a solution of chloride of magnesium, which gives a solution containing chloride of nickel."

Zinc.

Zinc from Iron and Zinc Sulphide Ores.—Elsewhere in this department, under the heading Nickel, a process of Diehl and Koehler is described, which may also be applied to winning zinc from sulphide ores, containing iron and zinc, the iron existing in the ore in the ferrous condition. Such an ore is first chloridized, to convert both iron and zinc into chlorides, and is then roasted in an oxidizing atmosphere at such a temperature that the chloride of iron is decomposed, but not the zinc chloride. The roasted ore contains ferric oxide and zinc chloride, of which the latter is extracted by leaching.

Alloys.

German Silver.—A. E. Hobson (846,851, March 12, 1907; assigned to International Silver Co.) patents the composition of a German silver alloy of fine appearance and very durable, and which may be freely worked and spun without liability of fire cracking. The alloy consists of copper, zinc, nickel and manganese, but the manganese should not be in excess of 4 per cent of the entire amount nor the zinc in excess of 20 per cent of the entire amount. The nickel and copper may vary, but a relatively high proportion of copper is generally used. Two suitable prescriptions are as follows: 67 per cent Cu, 20 Zn, 10 Ni and 3 Mn, or 59 per cent Cu, 20 Zn, 18 Ni and 3 Mn.

Mercury.

Mercury Retort and Furnace.—C. F. Brown (847,399, March 19, 1907) patents a mercury furnace consisting of a long inclined chute, through which the mercury ore, supplied at the top, slides down. Below this ore chute is provided a flue for the ascending heating gases, so that the bottom of the ore chute is at the same time the top of the gas flue. The ore passing down in the chute is heated by the gases passing upwards in the flue. The mercury vapor is carried off through a passage at the top of the ore chute. In order to assist the carrying off of the mercury vapor the following arrangement is made: The heating gases in their upward passage through the flue meet at intervals certain abutments, and at these places passage is provided for part of the heating gases to enter into the ore chute, where they mingle with the descending ore and help carrying off the mercury vapor to the exit at the top, while at the same time the ore is dried.

Mercury Condenser.—The same inventor (847,547, March 19, 1907) has devised condensing apparatus for the mercury vapor, consisting of a series of chambers. Partitions are provided in each chamber so as to fill partly the cross-section in such a way that the mercury vapor in passing from one end of the chamber to the other end is forced to pass along a zig-zag path. Near the top of each chamber water pipes are provided so that the hot vapors which ascend to the top are cooled and the mercury is condensed and drops down to the

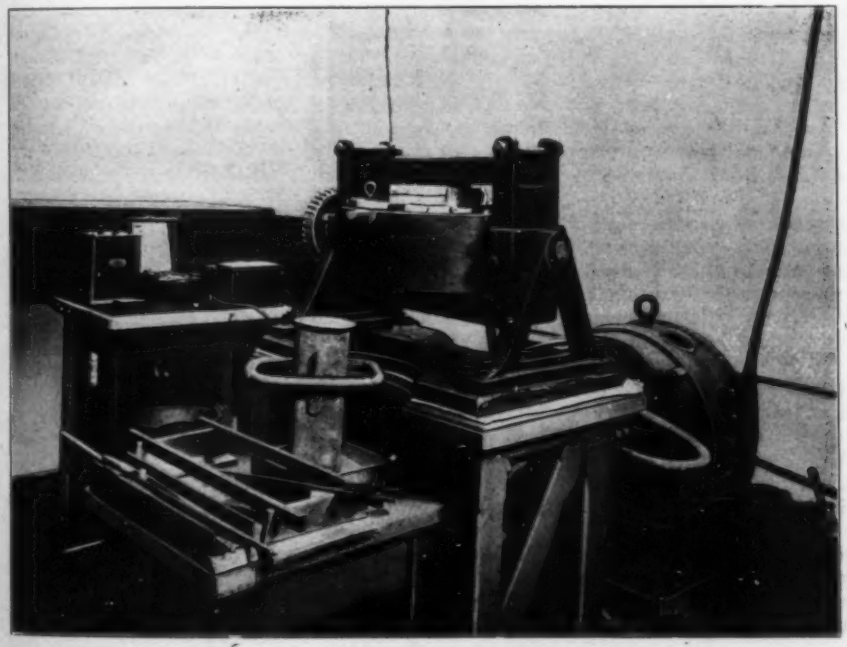
bottom of the compartments, and from there into troughs sloping down from one side of the chamber to the other, from which the mercury can be drawn off at intervals. The inner brick walls of the condensing chambers are formed with a glazed surface, preventing the penetration of the mercury vapor into the brick. Since in the condensation of the mercury minute particles adhere to all the interior surfaces, the interior is thoroughly washed at intervals, so as to remove these particles. For this purpose apertures are provided near the top through which a hose may be introduced.

Laboratory Induction Furnace.

The following details refer to a Kjellin induction furnace for laboratory purposes, which was exhibited at Sheffield during October last, and which presents some interesting features. The adjoining illustration gives a view of the furnace.

The furnace has a maximum capacity of 15 kw., and is designed for a frequency of 150 to 200 cycles per second. The furnace has been specially built for this high frequency to suit the conditions at the laboratory in which it was to be installed after the exhibition. It is, of course, well understood that the design of an electric induction furnace is rendered very much more difficult by use of a high frequency, and for this reason the behavior of the furnace, as given in the record of tests below, is remarkably good and the power factor very satisfactory, in view of the high frequency and the comparatively small size of furnace.

The furnace was originally intended for melting platinum, but at the exhibition was used for melting steel, and for this purpose it proved a thorough success, several castings being made on several days of the exhibition. A furnace of the size exhibited will pour an ingot of about 30 pounds.



KJELLIN TILTING INDUCTION FURNACE.

The furnace is built as a tilting furnace, just like the Kjellin furnace now in industrial operation for steel refining in the steel works at Voelklingen, as described and illustrated in our last issue, page 92. It is, of course, easy to so change the design as to use currents of a lower frequency than 150 to 200 cycles.

The lining is made of magnesite and gave no trouble.

The tests, of which a record is given below, were made on the afternoon of Saturday, Oct. 20, 1906:

Second Run, 200 Cycles.

Time.	*Volts.*	*Amps.*	*Watts.*	*Power Factor.*
6.00	195	39	4,320	....
6.03	205	40	4,700	0.57
6.08	220	35	4,080	....
6.14	225	36	4,600	....
6.16	215	40	5,200	0.605
6.18	215	40	5,040	....
6.20	210	41	5,310	0.62
6.22	teemed			

Third Run, 200 Cycles.

Time.	*Volts.*	*Amps.*	*Watts.*	*Power Factor.*
8.00	205	40	....	....
8.15	207	40	....	....
8.17	212	38	4,200	....
8.20	207	40	5,030	0.61
8.25	213	38.5	4,800	0.60
8.27	...	43	....	....

The Kjellin induction furnace is built in this country by the American Gröndal-Kjellin Co., of 45 Wall Street, New York City, and we hope to report on some interesting developments on a commercial scale in one of our next issues.

Ball Filling Material for Sulphuric Acid Towers.

In our Vol. IV., p. 466, we described the machine-made, perforated, Guttmann hollow earthenware balls of the Deutsche Steinzeugwaarenfabrik, of Friedrichsfeld, which are extremely suitable as a filling material for sulphuric-acid towers.

In a recent issue of the *Chemical Trade Journal*, Mr. Rudolf Heinz gives the results of an English installation of these balls.

A set of chambers of a capacity of 96,000 cubic feet, working on spent oxide, was provided with a Gay-Lussac tower, 8 feet square and 40 feet high, and 4,560 pounds of sulphur were burnt daily. It was only with great difficulty that the charges could be maintained at this quantity, and the Gay-Lussac tower was insufficient for the work. Two towers were then inserted into the set; firstly, a round intermediate tower between the second and third chambers, 6 feet in diameter and 13 feet 9 inches high, filled with about 10,500 Guttmann's patent 4-inch balls, and, secondly, a round preliminary Gay-Lussac, 5 feet 6 inches in diameter, and 11 feet high, filled with about 6,500 of these balls. The result of this change was very remarkable indeed. The charges could be increased by 23.16 per cent, so that now 5,616 pounds of sulphur are burnt daily. The small preliminary Gay-Lussac tower, of which only 166 cubic feet were filled with balls, does almost the whole of the work, the gases issuing therefrom being nearly colorless.

The analysis of these gases showed a total acidity of 3.52 grains per cubic foot at the inlet, and 2.30 grains at the outlet of the preliminary Gay-Lussac tower. Crowder, in the *Jour-*

nal of the Society of Chemical Industry, 1891, p. 303, gives the following data:

	Total Acidity.	
	At the Inlet of the Gay-Lussac.	*At the Outlet of the Gay-Lussac.*
New dust kilns	3.54	2.03
Lump ore kilns	3.55	1.80

It is thus seen that the little ball Gay-Lussac has really been doing almost the whole of the work that is generally demanded from the usual large Gay-Lussac tower, and the effect in future will be that sulphuric acid works will be able to dispense with these costly and cumbersome pieces of apparatus.

The Deutsche Steinzeugwaarenfabrik as manufacturer of the Guttmann hollow balls, is represented in this country by Messrs. Fred. Bertuch & Co., of New York City.

Zinc and Lead in Upper Mississippi Valley.

An old and famous mining region in which certain fundamental notions of ore deposit were first worked out is that part of the Upper Mississippi Valley celebrated for its zinc and lead mines. The boundaries of the region are in part indefinite, since sporadic occurrences of the minerals are found outside the mining region proper, but it is usual to define the area as including Grant, Lafayette, and Iowa Counties in Wisconsin, Jo Daviess County in Illinois, and an irregular narrow belt of territory in Clayton and Dubuque Counties paralleling the Mississippi River in Iowa. Dubuque, Galena, Platteville, and Mineral Point are the leading cities.

One man can do little toward solving the problems of such an area. It must be the work of many, and perhaps of years. This region has in fact been the scene of labor of some of our keenest students of ore deposits. A recent publication (Bulletin No. 294) of the United States Geological Survey, which is entitled "Zinc and lead Deposits of the Upper Mississippi Valley," by Mr. H. Foster Bain, now State Geologist of Illinois, will therefore be of interest to many students. It is hoped also that it may prove serviceable in the development of the region.

Mr. Bain's paper is primarily an account of the present condition of the district and a statement of ideas relating to the formation of the ores. It includes the results of observations, which began in 1899, when he was a member of the Iowa Geological Survey, and together with Professor Samuel Calvin made a study of the geology and mining industry of Dubuque County.

In 1893 Mr. Bain studied the mines of Jo Daviess County, Ill., for the Federal Survey and in 1904 the work was extended to cover the Wisconsin mines. As mining developments in this district have recently been very rapid, the publication of Mr. Bain's paper, which takes account not merely of the scientific problems presented in this area, but of its industrial growth as well, is especially timely.

It is estimated that from 1821 to 1904 the total lead production of the district was 611,975 tons, which sold for approximately $50,000,000. The maximum production was between 1840 and 1845. In 1845 and again in 1847 more than 27,000 tons of lead were shipped.

The total production of zinc up to 1905 is estimated at 450,000 tons, which probably did not bring much over $10,000,000. The early prices paid for the ore were very low—$3 to $8 for the carbonate and $4 to $12 for the blende.

In recent years the product has been entirely blende, which has sold for much better prices. The zinc production of the district is now rapidly increasing, and for the year 1905 approximated 33,000 tons. Many mines have been opened or equipped since the survey, upon which this report is based, was made. Mr. Bain not only describes most of the mines of the district individually, but explains the methods of prospecting, mining, and milling here in vogue.

The scientific conclusion of Mr. Bain's consideration of this area may be summarized as follows: "The ores of this region are believed to represent concentrations of material originally disseminated through the surrounding rocks, particularly the Galena and upper Platteville beds. It is believed that the zinc and lead originally present in the crystalline rocks of the Lake Superior district were delivered by rivers to an especially favorable portion of the Ordovician coast, where in warm, shallow waters they were partially concentrated by evaporation and were locally precipitated in quantity.

"The agents of precipitation were hydrocarbons or hydrogen sulphide, or both, derived from decaying masses of algae of a peculiar type irregularly accumulated within certain shallow basins on the sea floor. The rising hydrocarbons continued to be effective throughout the period of deposition of the Galena dolomite.

"As the algae matter became compressed the beds above broke and settled, leaving open spaces of peculiar form for the ultimate reception of the ore bodies. The beds were slightly deformed, whereby the basins were accentuated and deep joint cracks were developed. The area was subjected to peneplanation and later to gorge cutting, whereby the overlying shale was cut away and a vigorus underground circulation was brought about. This circulation resulted in a concentration of the disseminated material, and with the fall of ground-water level the ores were successively reconcentrated until the present type was produced."

Brick for Cupola Linings.

The Harbison-Walker Refractories Co., of Pittsburg, has recently issued an exceedingly interesting and useful pamphlet on the best kinds of brick for cupola linings and on rules to be observed in laying the brick. In view of the enormous amount of experience accumulated by the Harbison-Walker Refractories Co, in many years of activity, the pamphlet should find the careful consideration of foundry superintendents. In the following we give in abstract the most important points:

There are three different classes of fire clays, viz.: soft clays, shale clays and flint clays. The soft and shale clays usually run high in silica and flint clays run high in alumina. Aluminous clays are more refractory than silicious clays, and they can be made into tougher and stronger brick, which is more suitable for use in cupola linings.

Blocks for lining around the bottom of the cupola and for a distance of 2 feet or so above the tuyeres are made by the Harbison-Walker Co. of flint clay high in alumina with sufficient bond clay to make the brick strong and tough. From a distance of about 2 feet above the tuyeres to the charging door, the blocks are made to stand abrasion as well as heat, and they are exceedingly dense, hard and strong.

The use of high-grade material means a greater first expense for the brick, but this is many times balanced in operation by the fact that stoppages for repairs and loss of output are reduced to a minimum. The use of more expensive high-grade material is thus easily seen to represent an important source of economy.

Of course, even with high-grade material it is necessary to avoid deterioration of a lining due to careless operation. One of the sources of failure is excessive pressure of blast, causing an impinging or cutting flame—something like that from a blow-pipe—to play on the brick work. Another reason is excessive slag, or slag of a very scouring nature, which cuts out the lining rapidly. Slag should be kept as neutral as possible. Limestone and oyster shells are the slag materials in most common use.

It is quite possible that the use of chrome brick in the lower part of the lining would prove an economy in the case of a cupola where firebrick cuts out very rapidly. "Chrome" brick will resist almost indefinitely the chemical action of scouring slags, and they are used in lining cupolas which burn basic

materials, such as dolomite and limestone, and for the bottom of soaking pit furnaces where the scouring iron slag cuts out the fire-clay brick.

The best material to be used for daubing is a plastic fire clay and silica. When properly mixed the mixture is plastic and adhesive, it does not crack, expand or contract and stands heat. The fire clay should be soaked for 24 hours or longer, as it absorbs water slowly. The use of split brick pressed into the daubing with the flat side against the lining is effective and is recommended. This, of course, decreases the amount of daubing to be used and prevents skin drying of same, also prevents breaking away from wall as soon as steam is generated behind the skin-dried surface. Split brick can be put in nearly as cheaply and quickly as a cupola can be daubed, and are almost equivalent to a new lining.

Some very useful pointers are given on brick-laying in order to get the best results as follows:

Brick should be laid in that portion of the setting for which they are intended to be used, as conditions vary in different parts of setting and brick are varied to meet conditions.

The clay used should always be of same grade as the brick that are laid with it. This is important, as one of the frequent sources of trouble is the use of inferior clay to lay brick.

Clay should be mixed to a thin soup and brick dipped in same, then rubbed to insure a brick-to-brick joint. Not more than 300 to 350 pounds of clay should be used with 1,000 brick.

All brick-laying should be as even as possible, so as to avoid projecting brick, which catch the flame. A perfect arch or circle over the fire will last almost indefinitely.

Heating up and cooling down slowly adds very much to the life of the brick work.

Cutting of brick should be avoided as much as possible, and regular 9-inch shapes, such as skew, arch, wedge, soap, split, key, etc., used instead of cutting 9-inch straights. A cut brick is never as good as a whole brick.

Notes.

American Electrochemical Society.—At the March meeting of the Board of Directors the following gentlemen were elected members of the Society: Messrs. R. D. Thomas, Oakmont, Pa.; C. H. Korr, Niagara Falls, N. Y., and S. O. Dillon, Bloomington, Ind. The names of the following gentlemen will come up for election at the April meeting: Messrs. William McMurtrie, New York City; Tracy D. Waring, Perth Amboy, N. J.; R. W. Lohman, Oakland, Cal.; H. B. North, Madison, Wis.; John L. Harper, Niagara Falls, N. Y. The preliminary programme of the Philadelphia meeting will be found on page 116.

Relative Cost of Power.—We intended to publish in this issue an abstract of Prof. Lucke's paper on the relative cost of power from water, oil, steam and gas, presented at the recent meeting of the New York section of the American Electrochemical Society. Since, however, Prof. Lucke will read a paper on the same subject at the general meeting of the Society in Philadelphia we reserve our report for that occasion.

American Foundrymen's Association and American Association of Brass Founders.—According to a recent circular letter, sent out by Dr. Richard Moldenke, secretary of the American Foundrymen's Association, the annual convention will be held in Philadelphia from May 21 to 23. A special meeting of the foundry, rolling mill, plating, supply and other interests connected with the brass industry will take place Wednesday, May 22, at 10 A. M., in the Second Regiment Armory, and it is hoped that the much-discussed organization of the brass interests along educational lines similar to the work of the American Foundrymen's Association may take place. Great efforts are being made to make the convention interesting and enjoyable, and it is expected that the attendance will be even greater than at the last convention in Cleveland, where all records were broken with the registration passing the 800 mark.

Sherardizing.—The American Galvanizing Co. has been formed to operate the Cowper-Coles process of "sherardizing" (a dry process for covering articles with zinc) in this country. The capital is all New York and Pittsburg money, and the New York end of the business is being handled by Messrs. H. W. Poor & Co., of 33 Wall Street.

An Electric Steel Plant in Canada.—A company has been formed in Canada, to be located near Niagara Falls, for the purpose of smelting iron ores and to produce steel and steel castings by the Hèroult electric-furnace processes. In Europe there are now four steel plants operating under the Héroult patents, and all now running are without exception a commercial success. The Canada plant is expected to be in operation by about August, 1907. It will also be used as a demonstrating plant to show the process to intending licensees. An experimental department will be added to make further experiments on smelting and treating all kinds of ores by the Héroult electric process.

The United States Steel Corporation's annual report, which has just been issued, shows the trust to be in excellent condition. The "net earnings" were nearly $25,000,000 in excess of the best previous year, 1902.

The Tin Products Co. is a New York corporation which has recently been formed to operate processes of detinning and producing various tin products. Mr. George F. Seward is the president. Among the directors are Messrs. C. E. Acker, Geo. O. Seward and J. A. Wilke. Mr. Acker will leave Niagara Falls and reside in future in New York City.

New Journals.—*Copper and Brass* is the title of a new monthly journal published by the Copper and Brass Publishing Co., of Detroit, Mich., Mr. R. Marshall being the editor. The journal is devoted to the interests of the copper, brass, bronze and aluminium industries. Among the reading matter in the first issue (March, 1907) is the first of a series of elementary articles by H. M. Lane, on sources of the non-ferrous metals; an article by Paul Leake, on the copper situation; notes on molding of odd castings, on the deoxidation of brass and bronze alloys, on aluminium in typewriters, etc. With the beginning of this year a new monthly electrochemical and electrometallurgical journal has made its appearance in France. Its title is *Revue de l'Electrochemie et l'Electrometallurgie.* It is published at 1 Boulevard Saint Germain, Paris, but neither the name of the publisher nor of the editor is given. It seems, however, that the journal is published under the auspices of M. Adolphe Minet. Among the articles published in the first two issues is a summary by Ad. Jouve, of electrochemical processes for the fixation of atmospheric nitrogen; an article by A. Minet on classification of and literature on electric furnaces; an anonymous article on units and constants of physical chemistry; the Academy paper, by H. Moissan and T. Watanabe, on the distillation of metallic alloys, etc. We extend to our new contemporaries our wishes for a useful and prosperous career.

Electroplaters' Voltmeter.—It has long been recognized that the highest efficiency cannot be obtained in an electroplating establishment without carefully regulating the electrical conditions by means of measuring instruments. But platers have hesitated to equip each tank with a voltmeter on account of the high cost. This drawback is completely overcome in an instrument which has just been placed on the market by the Weston Electrical Instrument Co., of Waverly Park, Newark, N. J. It obviates the necessity of having an instrument for each tank and enables the plater to avail himself of an extremely accurate and reliable instrument at a very low cost. The instrument is really a combination, in one case, of a

voltmeter with a switch, by means of which the voltmeter can be connected to any one of a number of plating tanks in an establishment. We intend to give a more detailed description of the instrument in a future issue.

The Kny-Scheerer Co., of New York City, Department of Laboratory Supplies, have sent us their voluminous catalogue on chemical apparatus and laboratory supplies in the field of general chemistry. It is a bound volume of 612 pages, giving descriptions, illustrations and prices of not less than 5,789 different apparatus and appliances carried in stock by this company. For convenience the names of the different apparatus are arranged alphabetically. The catalogue is issued at present in German, but it is the intention of the Kny-Scheerer Co. to issue it in English at a later date. This company is a consolidation of Vereinigte Fabriken für Laboratoriums-Bedarf, Berlin; Max Kaehler & Martini, Berlin, and Dr. Peters & Rost, Berlin; The Kny-Scheerer Co., New York; The Laboratory and School Supply Co., New York.

Transportable Batteries.—We have received from the General Storage Battery Co., of New York City, a copy of the instructions for the installation, treatment and maintenance of transportable batteries made by this company. The instructions are very clear and to the point.

The Buffalo Dental Manufacturing Co., of Buffalo, N. Y., have recently issued their profusely illustrated catalogue B, list No. 34, on laboratory and workshop appliances, and especially laboratory furnaces, burning gas, gasoline and kerosene, blow-pipes, etc., for colleges, schools, chemists, assayers, jewelers, experimental laboratories, workships, etc.

Cement Making Machinery.—Catalogue 7 of the Power & Mining Machinery Co., of Milwaukee, gives a well-written outline of the methods of manufacturing Portland cement, with very neatly illustrated descriptions of all kinds of machinery used in this industry.

Fire-Proof Materials.—We have received from Messrs. Wendell & MacDuffe, New York City, a pamphlet on their asbestos and cement fire-proof materials. The line includes reinforced asbestos corrugated sheathing, a strong and durable composition of asbestos and cement compressed under strong pressure and reinforced with ¾ inch of woven wire mesh. This material is used as a substitute for corrugated iron, the particular advantages being its lightness, its lasting quality and the fact that it requires no painting whatever. The pamphlet also describes the asbestos building lumber, which is used to a great extent for construction work in electrical plants of all kinds for insulations, door panels, etc.

Modern Plate Glass Plant.—The Crystal City plant of the Pittsburg Plate Glass Co. is now about completed at Crystal City, Mo., 28 miles below St. Louis. There are fifteen buildings of reinforced concrete. It is estimated that 50,000 barrels of cement will be used in the building process, the brand being the "Universal Portland," made in the cement division of the United States Steel Corporation. Even the roofs are to be reinforced concrete tile, 4 feet x 8 feet. The machinery will be driven by electric motors of exceptional size. The power equipment already contracted for includes an 1,800 Allis-Chalmers gas engine, direct connected to a 1,000-kw. Allis-Chalmers generator. This unit will be installed in the same power house with a 5,000-hp. Allis-Chalmers "Big Reliable" engine, which carried the entire lighting load for the illuminating of buildings and grounds of the Louisiana Purchase Exposition at St. Louis; 1,600 hp. in Allis-Chalmers induction motors will be used to drive grinders and polishers.

Austrian Magnesite.—A consular report from Vienna states that the very considerable export of calcined magnesite from Austria to the United States, amounting during 1906 to nearly 53,000 tons net weight from the Vienna consular district alone has attracted general attention and that new magnesite mines are being located in several parts of Austria.

Digest of U. S. Patents.

Compiled by Barnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Calcium Carbid.

No. 492,377, Feb. 21, 1893, Thomas L. Willson, of Leaksville, N. C.

Refractory ores or compounds, such as those of barium, calcium, magnesium, titanium, tungsten, zirconium, silicon and boron are smelted in an electric arc furnace. The process, which is specifically described for the production of aluminum, aluminum carbid or aluminum bronze is said to have been used for the reduction of calcium oxid and production of calcium carbid. The furnace shown is a carbon or graphite crucible set in a fire-brick shell and serving as one electrode, preferably the positive one. It rests on a terminal iron plate which is bolted to the dynamo main. An adjustable carbon rod electrode depends into the crucible, which is covered by a plate, or two plates of carbon. An arc is sprung between the crucible and carbon rod and a mixture of the alumina or other compound, mixed with finely divided carbon, is introduced through a notch in the top edge of the brick-work, falling around the lower end of the rod, whereupon reduction proceeds. The rod is raised and kept at such height as to maintain the arc. Enough carbon is used in the charge mixture to prevent the formation of a fused bath, consequently there is no ebullition in the crucible. The alumina may be first introduced and fused and as it reaches ebullition the requisite amount of comminuted carbon be introduced, which then becomes commingled with the alumina by the violent circulation suppressing the ebullition. Ffteen per cent by weight of carbon is sufficient to prevent the formation of a fused bath, but if the material is coarse or not intimately commingled, a larger proportion may be necessary. The charge mixture may be made by drying, coking and powdering alumina impregnated with tar. Two modes of recovering aluminum are described: First, by maintaining a pool of molten copper or other alloying metal in the bottom of the crucible. Portions of the copat frequent intervals in small quantities. The mixture of alumina and carbon floats upon the bath of aluminum bronze per, alumina and carbon are preferably introduced alternately as a quiescent mass. Second, by employing such excess of carbon in the charge as to produce aluminum carbid, from which the metal may be subsequently abstracted. The product may be tapped out of the furnace through a hole closed by a plug of aluminum or clay. The arc is maintained close over that portion of the charge which is undergoing reduction. Two carbon-rod electrodes may be employed, with an arc sprung between their lower ends, or with two arcs in series between their lower ends and the charge.

No. 541,137, June 18, 1895, Thomas L. Wilison, of New York, N. Y.

Produces crystalline calcium carbid by electrically smelting a mixture of finely divided coke 35 per cent, and lime 65 per cent. The furnace comprises a wall of brickwork inclosing a carbon crucible, seated on a metal plate which is connected to one pole of an alternating-current dynamo. A layer of broken carbon is placed in the bottom of the crucible. A tap-hole extends through the lower end of the crucible and the brickwork. The other electrode is a depending carbon rod, adjustably supported. The dynamo supplies alternating current at a potential of 55 volts. The amperage for an electrode measuring 8 inches on the side may be 1,500. The molten carbid may be tapped out or may be allowed to accumulate in the crucible to a height of 2 feet or more, itself constituting the lower electrode. The alternating current at fifty reversals a minute assists the feeding of the pulverized charge into the arc by producing a series of explosions, and the yield per electric horse-power is almost double that of a process using direct current.

NEW BOOKS.

Electrochemistry.—First Part: Theoretical electrochemistry and its physico-chemical foundations. By Dr. Heinrich Danneel. Translated from the German by Edmund S. Merriam, Ph. D. 181 pages, 18 illustrations. Bound in cloth. Price, $1.25 net. New York: John Wiley & Sons.

Introduction to Physical Chemistry.—By James Walker. Fourth edition. 387 pages, illustrated. Price, $3.25 net. New York: The Macmillan Co.

Chemical Experiments.—Prepared to accompany Remsen's "Introduction to the Study of Chemistry." By Ira Remsen. Third edition revised by John Elliott Gilpin, 160 pages, illustrated. Price, 50 cents. New York: Henry Holt & Co.

Introduction to Metallurgical Chemistry.—For technical students. By J. H. Stansbie. Second edition. 252 pages, 45 illustrations. Bound in cloth. Price, $1.25 net. New York: Longmans, Green & Co.

The Manufacture and Properties of Iron and Steel.—By Harry Huse Campbell. 659 pages. Price, $5. New York: Hill Publishing Co.

Gold Mining Machinery.—Its selection, arrangement and installation. By W. H. Tinney. 308 pages, illustrated. Price, $5 net. New York: D. Van Nostrand Co.

Elements of Mining, Geology and Metallurgy.—A practical field and office manual, reference compendium, treating on the geology of mining and metallurgical methods of Western America. By G. Washington Miller. 489 pages, illustrated, folded plate diagrams. Price, $5.50. Denver, Col.: *Daily Mining Record.*

Principles and Practice of Coal Mining.—By James Tonge. 364 pages, illustrated. Price, $1.60 net. New York: The Macmillan Co.

Mining Law in Practice.—A brief and comprehensive treatise with geometrical illustrations of mining rights and correct methods of locating, holding and acquiring patents to United States mineral lands, according to Federal statutes and common practice. By G. Washington Miller. 292 pages, illustrated. Price, $2. Denver, Col.; Ores & Metals Publishing Co.

Steam Turbines.—Theory and Practice. By Lestre G. French. 418 pages, many illustrations. Battleboro, Vt.: *The Technical Press.*

Henley's Twentieth Century Book of Recipes, Formulas and Processes.—Containing nearly ten thousand selected scientific, chemical, technical and household recipes, formulas and processes for use in the laboratory, the office, the workshop and in the home. Edited by Gardner D. Hiscox, M. E. 787 pages, illustrated. Bound in cloth. Price, $3. New York: The Norman W. Henly Publishing Co.

German Monographs on Applied Electrochemistry.—Vol. XXVI. Elektrometallurgie des Eisens. By Prof. Dr. Bernhard Neumann. With 89 illustrations; paper cover. Price marks 7 (in New York, $2.35). Halle a. S.: W. Knapp.

German Monographs on Applied Electrochemistry.—Vol. XXVII. Die Elektrolytische Gewinnung von Brom und Jod. By Dr. Max Schlötter. 50 pages; 18 illustrations; paper cover. Price, marks 2.40 (retail price in New York, 80 cents). Halle a. S.: Wilhelm Knapp.

A Text Book of Electrochemistry.—By Max LeBlanc. Translated from the fourth enlarged German edition by Willis R. Whitney, Ph. D., and John W. Brown, Ph. D. 338 pages, 47 illustrations; bound in cloth. Price, $3.60 net. New York: MacMillan Co.

Census of Manufactures: 1905—Petroleum Refining.—Department of Commerce and Labor, Bureau of the Census Bulletin 70. 57 pages, illustrated. (A review, by Dr. Chas. E. Munroe, of the industry, with full statistical data. As appendix a bibliography is given with an exceedingly useful Digest of United States patents relating to petroleum refining.)

BOOK REVIEWS.

Handbook of Metallurgy. Vol. II.; Zinc, Cadmium, Mercury, Bismuth, Tin, Antimony, Arsenic, Nickel, Cobalt, Platinum, Aluminium. By Dr. Carl Schnabel. Second English edition. Translated by Henry Louis, M. A. 867 pages, 532 illustrations. Price, $6.50 net. London and New York: The Macmillan Co.

This book completes the second English edition of this well-known work. The title is a misnomer, since (with Vol. I. treating of copper, lead, gold and silver) the treatise is not upon the processes of metallurgy in general, it does not touch on iron or steel at all, and would be properly entitled a treatise on the metallurgy of the non-ferrous metals.

While the work cannot compare in completeness with the special treatise on the separate metals, such as Peters' Copper Smelting, Hofman's Lead, Eissler's Silver and Gold, Ingall's Zinc and Richards' Aluminium, yet on each of the metals just mentioned we are inclined to put Schnabel's discussions in the second place, and regarding the metals, not the subject of these special treatises, we may put Schnabel in the first place.

The 115 pages on mercury needs only the description of the Dennis furnace (our Vol. IV., pp. 104, 463) to make it complete; the 78 pages on tin are the most satisfactory to be found anywhere; the same may be said of the 47 pages on antimony, the 28 pages on arsenic, the 127 pages on nickel, the 29 pages on bismuth, the 19 pages on cobalt and the 17 pages on platinum, although it could have been wished that the latter included the metallurgy of the other platinum-group metals. The 317 pages on zinc are splendidly written, and the 54 pages on aluminium is a great improvement on the chapter in the first edition, but is yet considerably below the standard of the rest of the book.

The book bears the earmarks, in places, of having been uncritically revised by the English translator. Such formulas as Al2/6O1/2, taken from a poorly written French article, without change, are an evidence of careless compilation. Prof. Louis has brought the translation well up to date, but his work is not so carefully or critically done as the original body of the treatise, written by Dr. Schnabel.

A suggestion of the kernel of a metallurgical library is to take Schnabel's Allgemeine Hüttenkunde (not yet translated into English) for general metallurgical processes and apparatus, add to it Ledebur's unequaled treatise on Eisenhüttenkunde for the metallurgy of iron (also not yet translated—they both should be), and finish with the two volumes of Schnabel, which Prof. Louis has made accessible to English readers.

The printer and publisher have done their part excellently.

* * *

The Corrosion and Protection of Metals. By A. Humboldt Sexton, F. I. C., F. C. S. Price, $2.00. Manchester: The Scientific Publishing Co.

This small book is composed of notes that "have been written to supply to some extent the need for information" on the important subject of the corrosion and protection of metals. "These notes do not claim to be complete, but rather to represent a preliminary study of the subject." The book certainly does not cover all the work that has been done on this subject, but we can well believe that "its preparation has involved no small amount of labor."

There can be no doubt that more attention must be devoted to this important subject, and it is not surprising that the author complains of the scanty literature on the subject. In this country we may hope that some valuable data may be obtained by the committee on preservative coatings for iron and steel of the American Society for Testing Materials. This committee has appointed various sub-committees, which presented interesting reports at the meeting held at Atlantic City in June, 1905. Prof. Sexton does not expect that the opinions

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expressed in his book will be unanimously accepted, and probably they will not. If the book, however, stimulates an interest in the subject it will serve a most useful end.

Some recent work done on the subject of corrosion and protection of metals is not mentioned in this book, and this we might call attention to were it not that there is no means of ascertaining when the book was written or published. The omission of the date of publication of books of this kind is irritating.

* * *

CONVERSATIONS ON CHEMISTRY. First Steps in Chemistry. By W. Ostwald.

Part I: General Chemistry; translated by Elizabeth Catharine Ramsey; 12mo., 250 pages, 46 figures. Price, $1.50.

Part II.: The Chemistry of the Most Important Elements and Compounds; translated by Stuart K. Turnbull; 12mo., 373 pages, 32 figures. Price, $2. New York: John Wiley & Sons.

"The causes which led me to write this work lie partly in the past, partly in the future. The former spring from the feeling of thankfulness with which I even now regard the 'Schule der Chemie' of Stöckhardt, whose memory still lingers among us. By a stroke of good fortune this excellent work was the first text-book of chemistry which was placed in my hands, and it influenced the whole of my subsequent activity in science. Owing to the carefully thought-out directness in representing the facts to the pupil, the skill in selecting experiments suitable to the physical and mental powers of the beginner, I have never lost touch with experiment, although I have been chiefly occupied with general questions of science. The request of the publishers, who used to issue this work, that I should write a modern Stöckhardt, was both an honor and an opportunity of paying off an old debt of thankfulness."

As regards the future, Ostwald points out that in the hasty development of industrial chemistry in the last decade, it has been almost entirely organic chemistry which has developed in the direction of the discovery of new bodies and their systematic arrangement. But inorganic chemistry was a science before organic chemistry was thought of. Moreover, the processes of inorganic chemistry form the basis of chemical technology, on which that of organic compounds is a superstructure; and the complaint has been made in manufacturing circles that the young chemist trained exclusively in organic chemistry is unfit to cope with the solution of general problems. Ostwald maintains that the remedy must come from general and physical chemistry. It deals with questions which lie at the basis of organic and inorganic, of pure and applied chemistry. It forms a foundation for all real chemical education, and must be regarded as lying at the root of all chemical teaching, especially in its earlier stages.

This fundamental doctrine of Ostwald is hardly disputable. Chemistry and physics are separate sciences only because men, looking in different directions, made them so. A natural phenomenon is a chemical or physical phenomenon according to whether the observer is a chemical or physical specialist. In either case the point of view is one sided. A full view of the phenomenon is obtained only from all sides, i. e., on the basis of physical chemistry. This is most important for industrial work. For the industrial engineer must try to imitate Nature as it is in its complexity, not as it is looked upon by the scientific specialist for special research work, tending in one direction. The earlier the young student understands this truism, the better will be his equipment for later industrial work.

Wilhelm Ostwald is the most brilliant leader of physical chemistry in the world to-day. After having written his voluminous handbooks for the use of his colleagues and advanced students, he now presents this little book to beginners for their first steps in chemistry. Every page shows that the book is a work of love. It also shows Ostwald as an artist.

Being written in the form of a dialogue between master and pupil it may appear to some as somewhat crude art (though admirers of Plato would strongly object to such a conclusion). But it is certain that Ostwald could in this way show more clearly his individual physico-chemical personality, than if the book had been written in form of an ordinary description. The whole arrangement of the matter, the way in which the different principles are introduced, shows Ostwald's individuality. For this reason, besides others, the book should appeal to advanced chemists and teachers as well as to beginners. "For, as I frankly say, it is not for beginners only that I have written this book."

Without Ostwald, physical chemistry would not be what it is to-day. He is responsible for the direction in which the newly united forces of chemists and physicists have proceeded. It is still an open question whether some physical or specifically some electrical viewpoint has not been emphasized too much in the progress which has been made, to the detriment of the chemistry of the subject. It is, therefore, an open question whether Ostwald has not gone too far in emphasizing, for instance, the ionic theory in this text-book for beginners. But we cannot say this without expressing our admiration on the immense amount of information which Ostwald has really embodied in the small space at his disposal, though he has most happily succeeded in preventing the idea that the student now knows everything. The book is full of suggestive hints which should early fill the student "with that longing to pry into the mysteries of science" which constitutes an essential qualification for his future usefulness to science and mankind.

The first volume deals with general chemistry and is divided into thirty chapters: substances, properties, substances and mixtures, solutions, melting and freezing, boiling and evaporation, measuring, density, forms, combustion, oxygen, compounds and constituents, elements, light metals, heavy metals, more about oxygen, hydrogen, oxygen and hydrogen, water, ice, steam, nitrogen, air, continuity and exactness, the expansion of air by heat, the water in the air, carbon, carbon monoxide, carbon dioxide, the sun.

The second volume deals with the chemistry of the most important elements. The first chapters deal with chlorine, preparation and properties, chlorine and water. Then follows a chapter on acids and bases, and the simple neutralization experiments are used as the foundation to introduce the quantitative stoichiometrical laws. The following chapters are: chemical equivalents, the weights of combination, multiple proportions, the atomic theory, the law of gaseous volumes, electrolysis, acids, salts, the oxygen compounds of chlorine, bromine, iodine, sulphur, sulphuric acid, hydrogen sulphide, nitrogen and nitric acid, ammonia, phosphorous, carbon, silicon, sodium, potassium and ammonium, calcium, barium, strontium, magnesium, aluminium, iron, manganese, chromium, cobalt and nickel, zinc, copper, lead, mercury, silver (photography), tin, gold and platinum.

Both translators may be congratulated on the excellent work they have done. It may be added that the book has also been translated or is being translated into Swedish, Russian and Dutch.

* * *

Lehrbuch der Elektrochemie. By Prof. Dr. Max Le Blanc. Fourth edition. 319 pages, 25 illustrations. Price, in paper cover, marks 6; bound, marks 7 (retail price in New York, $2.00 and $2.35 respectively). Leipzig: Oskar Leiner.

This book is known as one of the best German books of small size on theoretical electrochemistry on the basis of the electrolytic dissociation theory and the theory of solution. Within eleven years after the appearance of the first edition three further German editions have become necessary, and the book has been translated into English, French and Italian. The author now occupies the chair formerly held by Wilhelm Ostwald in Leipzig.

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* * *

Some Founders of the Chemical Industry.—Men to be remembered. By J. Fenwick Allen. 289 pages, 15 portraits. Price, 5s. net (price in New York, $2.00 net). Manchester: Sherratt & Hughes.

The eight biographical sketches of founders of the alkali industry in Lancashire which are collected in this volume, were first published in the *Chemical Trade Journal.* The subjects of the eight sketches are William Gossage, Josias Christopher Gamble, James Muspratt, Andreas Kurtz, Henry Deacon, James Shanks, Christian Allhusen, Peter Spence.

These men were born between 1776 and 1822, they died between 1848 and 1890. We have, therefore, here to do with the history of a past epoch. This is also true in another sense. The story of the life and work of these men is an outline of the individualistic period of the development and growth of the Lancashire alkali trade. Since 1890 the situation has completely changed. This trade has been transformed by the absorption of nearly all the private concerns into the great trust or combine—the United Alkali Co.

It is but natural that there is a decidedly personal element in the stories of this book. These eight men are well worth being remembered and the author has told their life well. The mechanical makeup of the volume is simple and decent.

* * *

Handbook of Mathematics. For engineers and engineering students. By J. Claudel. From the seventh French edition. Translated and edited by Otis Allen Kenyon. 708 pages, 422 illustrations. Price, $3.50. New York: McGraw Publishing Co.

This book is unique in several respects. It covers in a single volume the whole field of mathematics, from the simplest operations on whole numbers to principles and applications of calculus. While print, illustrations and general make-up are excellent the price of the book is exceedingly low. Evidently the publishers expect a very large sale and the book certainly deserves it.

The book has six parts: arithmetic (whole numbers, fractions, powers and roots, ratios, proportions and progressions, logarithms); algebra (fundamental operations, powers and roots of algebraic quantities, equations of the first, second and third degree); geometry (plane and solid); trigonometry (plane and spherical); analytical geometry; calculus (differential and integral calculus with applications).

Aside from its value as a text-book for the class room, the book should be extremely useful as a reference book. Probably every engineer or professional or commercial man in general will agree with the translator that "the use of text-books for reference is discouraging. For example, if a busy man wishes to solve an integral, which is not given in the table, he naturally refers to his college text-book on integral calculus, spends several hours studying, and finds his trouble is farther back, most likely in algebra; then the chances are that, due to lack of time, he will give up and declare that he has forgotten his calculus." In the present volume this trouble has been anticipated by the very frequent use of cross references, completely interconnecting all parts of the book. There is no index.

A special word of praise is due to Mr. O. A. Kenyon. His translation of the French text has faithfully preserved the conciseness, elegance and gracefulness which are so characteristic of the French school of mathematics. But Mr. Kenyon has been equally successful as editor in adapting the French book to the needs of American engineers and engineering students.